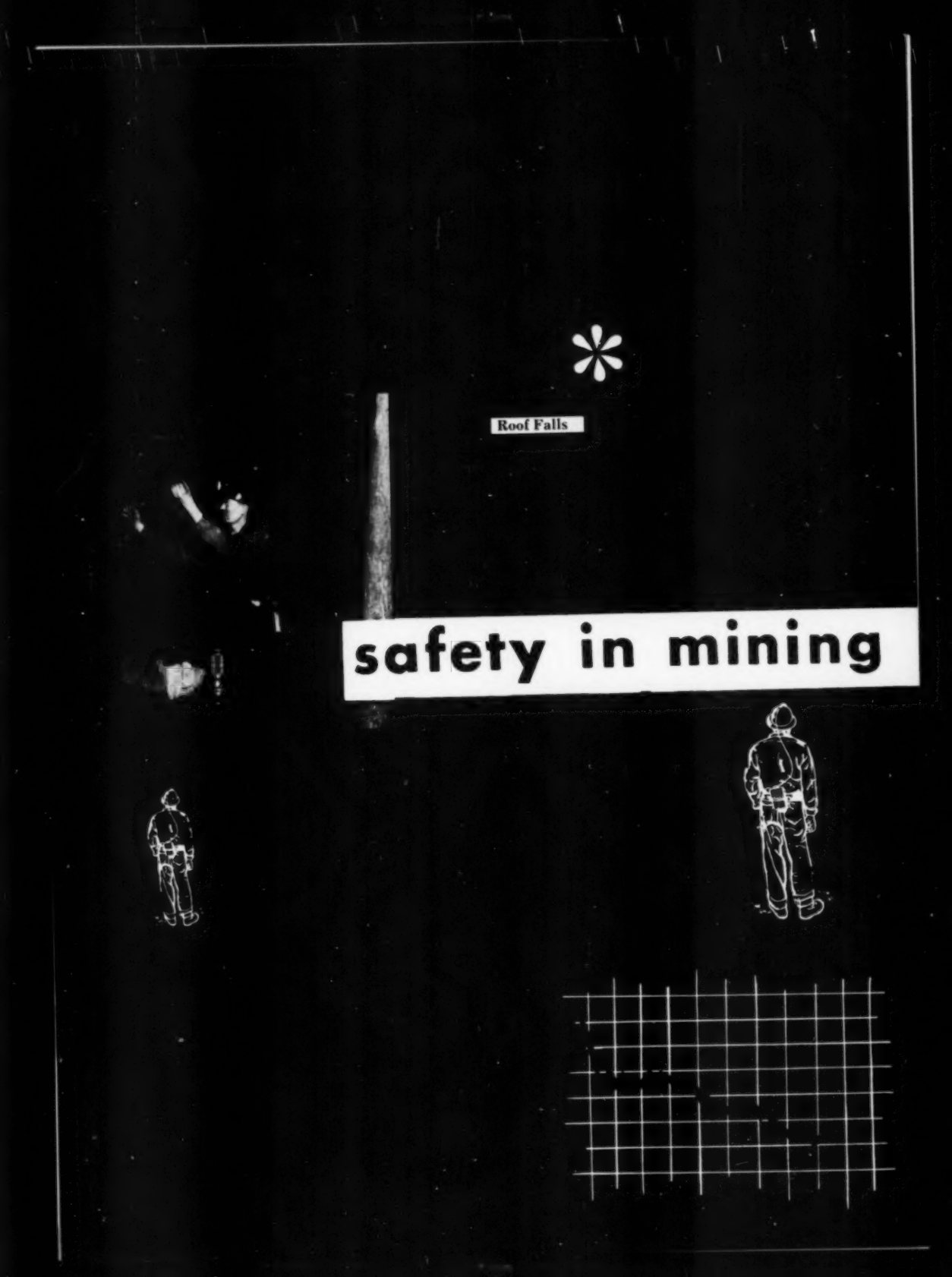
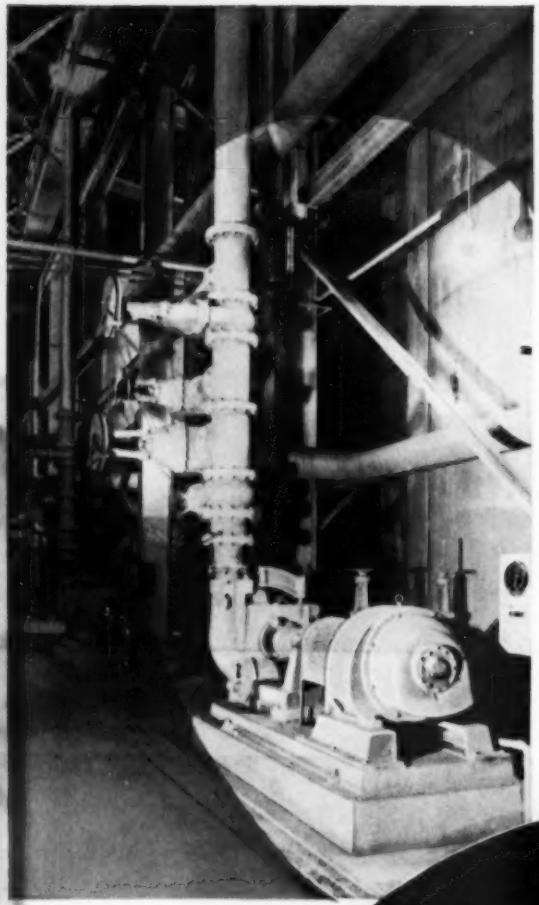

Roof Falls
safety in mining



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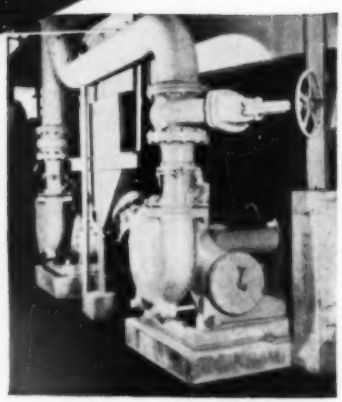

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MINING ENGINEERING

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VOL. 4 NO. 3 MARCH, 1952

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COVER

Symbolic drawing, showing the symbols of safety, the mine inspector and the fatality-rate chart and typical causes of underground accidents.

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General Motors Diesel
Case History 5012-12

USER: United Feldspar Minerals Corp.
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INSTALLATION: 5-year-old GM 4-71 Diesel (used previously on sawmill) replaces steam engine on Ingersoll-Rand FR-1 compressor.

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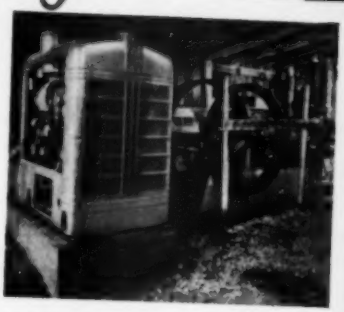

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Letters to the Editor

Hoisting Capacities

Without having asked for it, the writer has apparently been drawn into the discussion regarding relative hoisting capacities which was started by Woodward Iron, egged on by Joe Haller of Cleveland-Cliffs [*Mining Engineering*, August p. 642] and Axel Kjelgaard [November p. 919], who took exception to Mr. Haller's remarks of tonnage hoisted and avers that they do better in Sweden. It does not appear to the writer that Mr. Haller was inclined to refer to the tonnage hoisted at Cleveland-Cliffs in a manner to imply that they claimed any hoisting record, but rather as a statement that they had hoisted a certain amount of ore during the year 1950. Mr. Haller has been in the mining game long enough to appreciate what Mr. Kjelgaard did not apparently take into consideration, that length of haul, shifts hoisted, nature of ore, method of loading and the many other factors which influence the hoisting cycle make a comparison of tonnages hoisted of two given mines almost an impossibility.

Mr. Cheyette sent you a copy of his letter to Bob Manegold relative to the capacities of the Orient #2 mine. There are several mines around the Country which constantly set up records about which little or nothing at all is ever heard. We cite the Snow Hill Coal Co. mine at Terre Haute, Ind., which has an automatic Nordberg hoist and which hoists 12 tons per trip or 700 tons per hr from 700 ft. This is a counterweighted installation and a complete cycle is made every 62 seconds or 58 trips per hr. Another mine, The Potash Company of America mine at Carlsbad, N. Mex., has a Nordberg automatic hoist, hoisting eight tons per trip from 1150 ft on a one minute cycle. The materials hoisted in these two mines are ideal for automatic hoisting—they are free flowing so that no hang up is anticipated and hoisting is continuous. Multiplying out the above figures you will find that it is possible to hoist 11,000 tons in two shifts at Snow Hill or over 3 million tons in 300 days per year and over 2 million tons at Potash.

Other mines have records which can be pointed to with pride. When a record for one year is established, it doesn't stand too long as here in America the ambition of all the men connected with the operation forces them on to larger tonnages.

T. C. WIEDENHOEFER
MANAGER MINE HOIST DEPARTMENT
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Big Operation

Mining Engineering, Jan. 1952, p. 74, Fig. 4—Graham Snyder mine, Eagle Picher Mining and Smelting Co., February 1950. Scale: ½ in. is *approximately 300 miles.*

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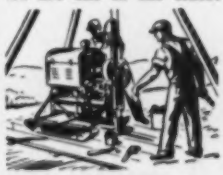

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MEET THE AUTHORS

S. H. ASH

S. R. B. COOKE

KARL V. LINDELL

S. H. Ash *(Mining Industry Continues with Lower Fatal-Injury Rates, P. 257)* was born in Roslyn, Wash. and attended Lehigh University and University of Washington. He received a B.S. and E.M. degree in mining engineering. Mr. Ash was supervising engineer in charge of work of Safety Div., Bureau of Mines, eastern Pennsylvania and the states of N. Y., Del., N. J. and the New England states. He was appointed chief engineer of Metal and Nonmetal Mining Section of the Mineral Production Security Div., Bureau of Mines, July 1942. At present he is chief engineer, Safety Branch, U. S. Bureau of Mines. Besides holding membership in AIME, he is a member of National Honorary Engineering Society, Society of American Military Engineers, Tau Beta Pi and many other societies.

S. R. B. Cooke *(Co-author with Cadwell and Philippoff, P. 283)* was born in Wanganui, New Zealand and attended Otago School of Mines, Missouri School of Mines, Carnegie Institute of Technology, and University of Missouri. He was a research metallurgist with Consolidated Mining & Smelting Co. of Canada. He was with state mining experiment station in Rolla, Mo. in cooperation with U. S. Bureau of Mines. Mr. Cooke was assistant professor of metallurgy and mineral dressing at Missouri School of Mines from 1939 to 1946. At present he is professor of metallurgy and mineral dressing at School of Mines, University of Minnesota. Besides being a member of AIME, he is associated with American Chemical Society, Astronomical Society of Pacific and British Astronomical Association.

Karl V. Lindell *(World's Largest Asbestos Producer Uses Block Caving and Concreted Slusher Drifts, P. 265)* is vice president of Canadian Johns-Manville Co., Ltd., and general manager of the Company's Asbestos Fibre Div., a position to which he was appointed on November 21, 1951. Mr. Lindell was born on November 14, 1905 in Ashtabula, Ohio, and was graduated from Michigan College of Mines in 1928 with a degree in mine engineering. His first position was with the Royal Tiger Mines in Colo. In 1929 he became associated with International Nickel Co. of Canada, Ltd., where he held positions of responsibility at the company's Frood mine and the Copper Cliff general headquarters in Northern Ontario. He was made assistant to the general superintendent of mines for International Nickel in 1943. In 1934 In-

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ternational Nickel sent Mr. Lindell to Petsamo, Finland, to develop a program for opening a nickel mine on which the company subsequently spent $7,000,000 only to have the property seized by the Russians in 1940, during the Russo-Finnish War. Mr. Lindell joined Canadian Johns-Manville in 1945 as mine superintendent at the Company's Jeffrey mine underground operations at Asbestos, Quebec. He was later appointed plant engineer for the Company's operations at Asbestos. In January 1950, he was made manager of the Jeffrey mine at Asbestos, Quebec. Active in civic affairs in Asbestos and an enthusiastic sportsman, Mr. Lindell is a member of the Knights of Columbus and the Asbestos Chamber of Commerce and a director of the Asbestos Hockey Club and Asbestos Golf Club. He is also a member of the Canadian Institute of Mining and Metallurgy, and the American Institute of Mining and Metallurgical Engineers.

D. E. Cadwell *(Contact Angles and Surface Coverage, P. 283)* was born in Minneapolis, Minn. and attended the University of Minnesota and has worked for the Consolidated Mining & Smelting Co. of Canada. At present time he holds position of junior testing engineer with this company. He is an AIME member and also holds membership in CIMM and Tau Beta Pi. His interests are music, books, photography and winter skiing.

When notifying AIME headquarters of a change of address, or of company position or affiliation, *please mention the Branch of the Institute to which you belong — Mining, Metals, or Petroleum.* This will make for a more expeditious handling of the change and will facilitate the preparation of various reports.

C. E. Lesher *(Development of the Disco Process of Low Temperature Carbonization, P. 287)* was born in La Junta, Colo. and attended the Colorado School of Mines. He was editor of Coal Age from 1920 to 1923. Mr. Lesher was vice president of Poughkeepsie Coal Company from 1927 to 1938 and from 1938 to 1950 was employed as president of Disco Co. At present he is a consulting engineer. He is an AIME member. Mr. Lesher received a distinguished achievement award from Colorado School of Mines in 1949.

W. Philippoff *(Co-author with Cadwell, P. 283)* was born in Peterhof, Russia and attended Technical High School in Berlin now called Technical University. He received a diploma in engineering in 1932 and a Doctor's degree in engineering in 1934. From 1932 to 1945 he worked for Kaiser Wilhelm Institute in Berlin. He was with Stanford University in California from 1948 to 1949. From 1949 to present he is with University of Minnesota. Mr. Philippoff is a member of American Chemical Society.

I. S. Tillotson *(Pipeline Transportation of Phosphate, P. 273)* was born in North Vasselboro, Maine and attended Rhode Island State College. He worked from October 19, 1923 to September 19, 1932 as a city engineer in Lakeland, Fla. He has been employed by International Minerals & Chemical Corp. as engineer and mining superintendent in Bartow, Fla. from September 1932 to date. Besides being a member of AIME, he holds membership in Florida Engineering Society and Lakeland Yacht Club. The study of military history is Mr. Tillotson's favorite pastime. This is the first paper he has presented before the AIME.

Glenn S. Wyman *(Primary Blasting Practice at Chuquicamata, P. 300)* was born in Richland, N. Y. and attended the Missouri School of Mines & Metallurgy. He received a B.S. in mining engineering in 1922. He has worked for Chile Exploration Co. from May 1922 to date except from May 1925 to December 1927 when he was employed by automobile dealer in Port Washington, L. I. Mr. Wyman holds membership in AIME and in Masonic Lodge. Golf and fishing are his favorite pastimes.

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A Statement by Anaconda on the Copper Situation

MANY users of copper have vital decisions to make . . . usually in connection with the present defense-induced shortages of copper and aluminum. This statement is an effort to remove the smoke screen surrounding the copper picture . . . to wipe away the confusion caused by too much talk supported by too few facts.

Substitution poses problems – Industry has been urged to substitute aluminum and other materials for copper. In some instances this may be logical and practicable. In many others it is difficult, if not impossible. But – before making *any* long-term decisions that may cost a great deal of money in engineering, new plant facilities or rescheduling of production operations – one should know the facts about the future of copper.

New Anaconda projects – The first major increase in copper production will come from Anaconda when the Greater Butte Project and the new Sulphide Plant at Chuquicamata, Chile, begin operations this spring. By 1953, these two projects should raise present levels of copper production by about 95,000 tons yearly.

Toward the close of 1953, Anaconda's new Yerington project in Nevada is expected to start producing at an annual rate of 30,000 tons. By then, Anaconda will be adding to the present yearly copper supply at the rate of about 125,000 tons.

Other new projects – During 1954-55 still other new projects in the U. S. and friendly foreign countries will further augment the increasing copper supply. All told, it is estimated that by 1955, not less than 450,000 tons of copper could be produced annually – over and above present production levels.

Accordingly, in 1955-56, domestic production plus imports could bring the U. S. copper supply to 1,800,000 tons yearly. This would represent an increase of about 20% over present levels. Based on historical comparisons, and barring a large-scale shooting war, this amount of copper could support a Federal Reserve Board Index of Industrial Production of 270, an increase of 24% over the present, and 45% above the first half of 1950.

• • •

These are the 'things to come' in copper. On the basis of the facts there is no necessity for considering long-range substitution of other materials for the red metal.

52338A

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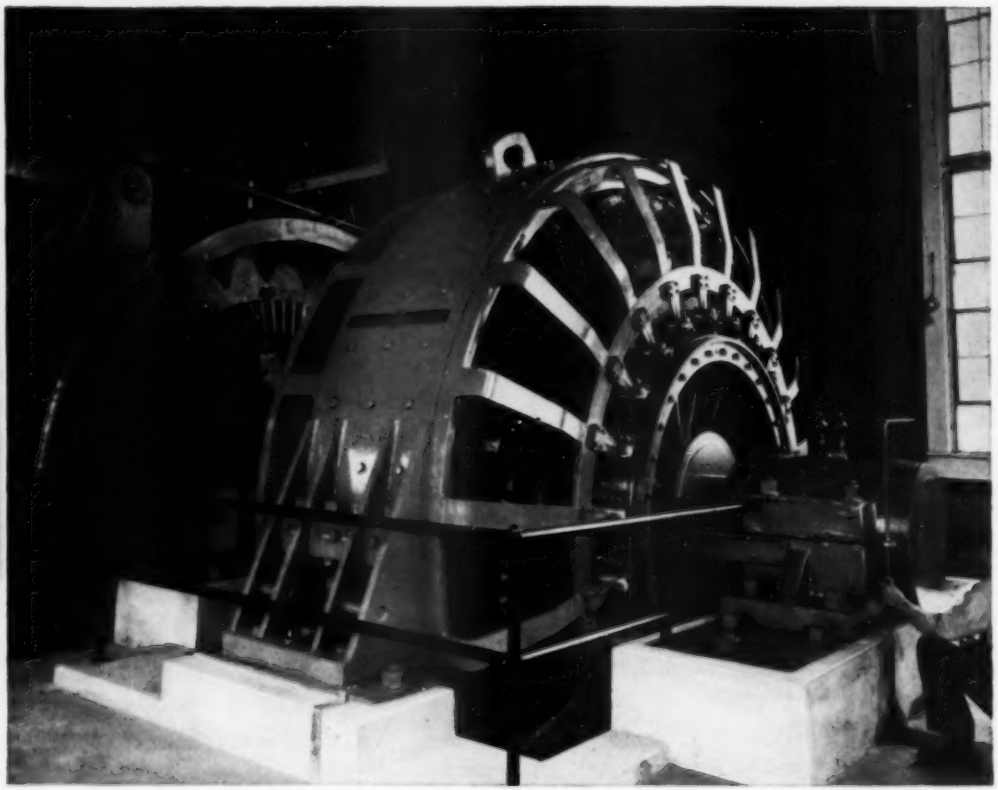
ANACONDA installs one of world's largest hoist motors

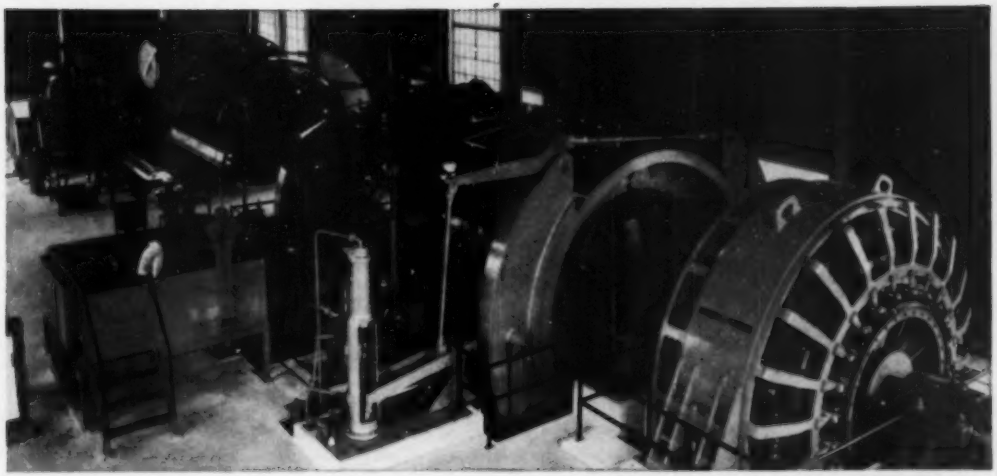
—for high-tonnage low-cost output!



Here it is installed—the G-E 3000-hp 600-volt 60-rpm d-c motor that will drive Anaconda's Kelley Shaft ore hoist. Scheduled to go into production in 1952, it will permit handling much larger tonnages than possible now. Skips, carrying 12 tons of ore per trip, will have a capacity of 853 tons per hour from 868 feet, with approximate power consumption of only 12.4 kwh per trip. From a 4335 foot depth, capacity will be 307 tons per hour, using about 57.3 kwh per trip. New hoist is expected to be able to handle approximately 10,000 tons of low-grade ore per day.

GENERAL ELECTRIC

660-21

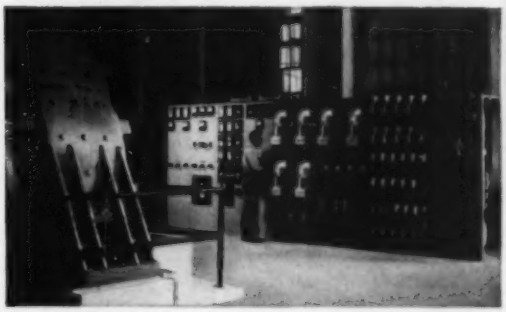


This over-all view shows the G-E 3000-hp motor direct-connected to the hoist's two drums, with the hoist operator at his control station. Expected capacity of the new hoist is 10,000 tons per day, supplemented by the 5000-tons-per-day capacity of the other hoist shown in background.

Complete G-E hoist-drive equipment for Kelley Shaft in Greater Butte Project includes 3000-hp d-c motor, permits lifting 12 tons of ore per trip at low kwh cost

High-tonnage low-cost hoisting is a "must" with Anaconda. Especially so on its Greater Butte Project for the long-range economical mining of an anticipated 130,000,000 tons of low-grade ore by block caving. That's why Anaconda, a satisfied customer since 1913, asked General Electric to supply the main ore-hoist drive in its Kelley shaft —including one of the largest single mine-hoist motors in the world!

General Electric's extensive experience covers over 900 large hoist-drive installations now serving the country's mines, helps explain why companies like Anaconda keep coming back for more of the same. When *you* call in a G-E mining-industry specialist, you put this experience to work for you —profitably. Meanwhile, send for Bulletin GET-1430, "Electric Equipment for Mine Hoists." *General Electric Company, Schenectady 5, N. Y.*



Control for the huge G-E motor is centered in this panel, built throughout to meet specifications laid down by Anaconda. Designed for either manual or automatic hoisting, the drive will operate two skips in balance, at a speed of 2250 feet—almost half a mile—per minute.

New G-E Motor Selection and Application Course can help train your mine and mill workers, increase their efficiency. A G-E "More Power to America" program, it shows how motors work, types in use, and how to select and apply them. Write now for Bulletin GEA-4938-16 describing the course in detail.

MINE-HOIST DRIVES

In addition to hoist motor and control panel, the G-E drive equipment includes this motor-generator set to supply the needed d-c power. It comprises a 2500-kw 600-volt d-c generator driven by a 3500-hp 514-rpm 2400-volt synchronous motor, and a 60-kw exciter.

Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Mining Engineer, 41, married, one child, employed. Broad experience both underground and open-pit type mining. Exploration, development, exploitation, industrial construction, milling in northwestern United States. Desires position with progressive, established concern. M-666-259-Chicago.

Recent Graduate, 24, married, no children, B.S. mineral economics. Employed—one year's experience in development of deposits, study of problem areas. Will accept position anywhere in the world. Available immediately. M-667.

Graduate Mining Engineer, 38, married, two children, Dutch nationality. Experience bauxite open-pit dragline, stripping and mining, nickel silicateous ore exploration, tin lode mining and dredging, ore dressing research, administrative work, all in responsible positions. Read, write and/or speak seven languages. Desires responsible position. Presently located Netherlands. M-668.

Mining Engineer, 33, veteran, married, employed. Three years' mechanized bituminous coal mine, one year metal mine. Surveying, mapping, conveyor-belt and shaft sinking experience. Desires progressive full time engineering work for coal mine or consulting firm under senior engineer(s). Any U.S.A. location. Available thirty days. M-669.

Mining Engineer, 34, married, B.S. degree. Excellent performance record in direction of exploration and development activities for base and precious metals. Well trained in large scale milling operations. Recent experience in field of raw materials for western steel industry. Will travel part time. Seeks management position. U. S. only. Available immediately. M-671-521-E-1-San Francisco.

Mining Executive with record of accomplishment organizing and directing efficient low cost mining and ore dressing operations available for position in the United States. Broad experience engineering, mechanized mining, flotation, safety, union labor relations and preparation of reports. Married, employed; available reasonable notice. M-672.

POSITIONS OPEN

Engineers. (a) Mill Shift Boss with flotation or cyanidation experience. Salary, $4800. (b) Mine Foreman, experienced working heavy ground. Salary, $5400 to $7800. (d) Mine Safety Engineer, experienced in ventilation. Salary, $6000. Permanent for right men. Location, Philippine Islands. S-1458.

Mining Engineer to take care of management and supervision of four small mines running from 75 tons per day up to 400 tons per day. Must be able to speak Spanish and get along well with native help. Salary open. Location, South America. Y6590.

Geologist experienced in structural interpretation, routine underground and surface mapping for gold vein mine. Salary, $6000 a year plus bonus of one month's salary. Knowledge of Spanish desirable. Location, Colombia, S. A. Y6581.

Assistant Editor, young, mining or metallurgical engineering graduate, with at least two years' experience in ore dressing, extractive metallurgy and allied fields, to prepare and edit articles covering mining metallurgy. Salary, $4000 to $5400 a year. Location, New York, N. Y. Y6577.

Engineers. (a) Mill Metallurgist, preferably with previous plant experience, as well as experience in testing laboratories. Should be capable of carrying out ore beneficiation tests and research work, and making reports on the results obtained. Work will be in the nature of laboratory testing, gravity concentration, flotation concentration and cyanidation and some roasting tests. Salary, $4200 to $4800 a year depending upon qualifications. Room and board and transportation, if single; and if married, furnished house, plus $75 a month board allowance after three months' single status. (b) Junior Mill Metallurgist with some laboratory experience, with knowledge of the technique to carry out assigned tasks with supervision. Salary, $3000 to $3300 a year, plus board allowance, etc. Location, Central America. Y6573.

Engineers. (a) Geological Assistant, woman, graduate, majoring in geology, familiar with determinative mineralogy and able to draw maps. Work includes ore estimating, library research, etc. Location, New York, N. Y. (b) Mining Engineer with operating experience in open-pit mining, familiar with surveying, pit layout, etc. Location, New England. (c) Metallurgist, experienced in ore dressing, including flotation. Work largely on special ore testing problems in Eastern states. Salaries open. Y6572.

Engineers. (a) Mining Engineer with open-pit and underground experience, to plan production layout and improvements on copper project. Salary, $5200 to $6500 a year. (b) Engineering Draftsman to do mapping, cross-sections, etc., covering work. Salary, $3900 to $4420 a year. Location, South America. Y6530.

Mining Engineer, young, with some experience, to perform the duties of a mine engineer and to assist the general superintendent with administrative and technical duties at a tin-tungsten operation. Should have some progressive underground experience qualifying him to supervise and train unskilled native labor, as well as to do ordinary mine surveying and geologic mapping. Single status during the current winter. Family accommodations later. Salary to start, $6000 a year plus living accommodations. Location, Alaska. Y6514.

Engineers. (a) Mining Engineer, 25 to 35, to assist chief engineer. Should have experience in mine surveying, with knowledge of drafting, surface plant and construction. Salary open. (b) Assistant Mine Superintendent, 30 to 40, for underground metal mine. Must have had previous underground experience in cut and fill, square set stopes and mechanical slushing. Salary open. Location, Caribbean. Y6415.

Engineers. (a) Assistant Chief Engineer with underground experience and preferably a working knowledge of Spanish. Salary, $4200 a year plus bonus. (b) Mill Superintendent with broad experience in milling. Should have working knowledge of Spanish. Salary, $5400 a year plus bonus. (c) Diamond Drill Foreman with knowledge of Spanish. Salary, $6000 a year. Single status for three months. Company will pay transportation. Living quarters and medical expenses provided for employee and family. Y6402.

Assistant Professor or Instructor. (a) Assistant Professor in mining engineering, 25 to 35, with some graduate work and practical experience, to teach mining engineering. Salary, $3600 to $4500 a year. (b) Instructor, with at least two years' practical experience, to teach mining engineering. Opportunity for graduate work. Salary, $3000 to $4000 a year. Location, Pennsylvania. Y6371.

Geologists, young, familiar with the coal industry. Location, Ohio. Y6484.

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USED BLUEPRINTING EQUIPMENT
IN GOOD CONDITION
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"Threads of Gold"
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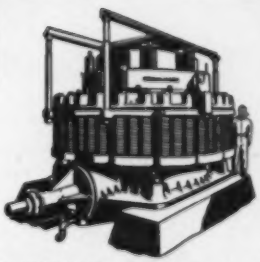
ROMAN TOGA

The mineral *Chrysotile,* meaning golden fibers, or threads, supplies over 90% of the world's asbestos requirements. Called the "miracle mineral" by the ancients, the Roman naturalist Pliny (23—79 A.D.) refers to the use of such material as "a rare and costly cloth" used for the funeral garments of kings.

The secret of the great usefulness of asbestos lies in its unique combination of fibrous structure, fire and water resistance, flexibility and strength. Since the profitable recovery of asbestos fiber depends upon fine crushing without destroying fiber length, a crusher that will perform such functions economically, and will produce large quantities of finely crushed material, is essential.

Proof of the world-wide acceptance of *"SYMONS"* Cone Crushers in asbestos production is found in the field, where practically every major producer employs these high capacity, economical machines . . . *as in all of the great ore and industrial mineral operations the world over.*



"SYMONS" Cone Crushers . . . the machines that revolutionized crushing practice . . . are built in Standard, Short Head, and Intermediate types, with crushing heads from 22 inches to 7 feet in diameter—in capacities from 6 to 900 tons per hour. C252

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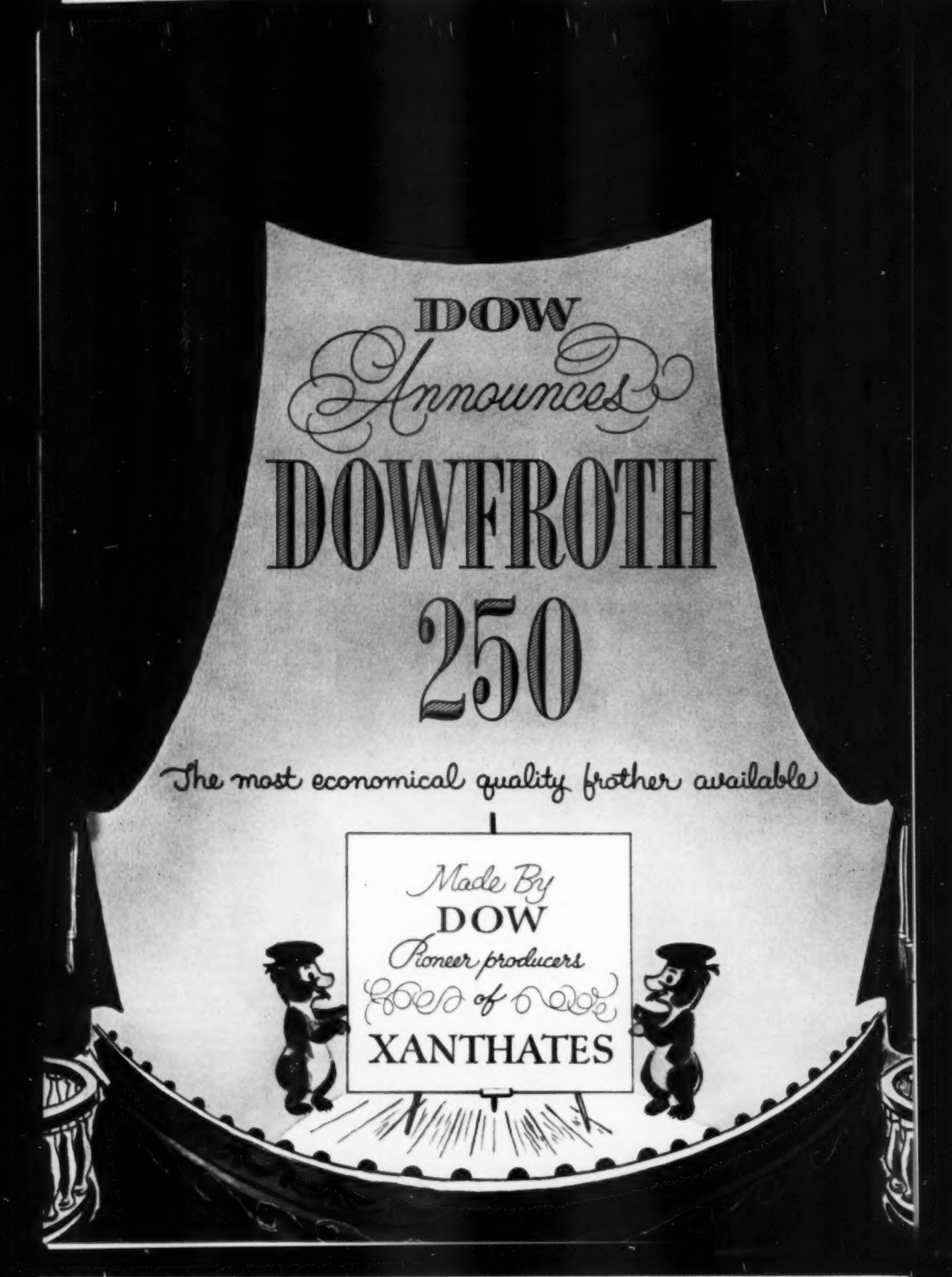
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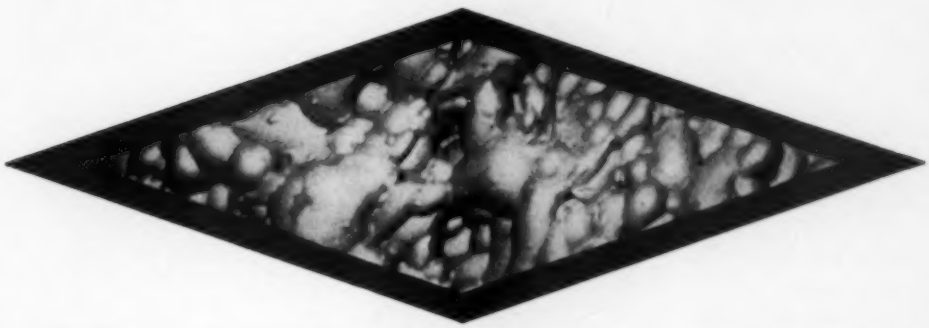

DOWFROTH 250 has demonstrated better frothing characteristics in actual mill tests where concentrations *as low as one-fourth of normal concentrations* have proved satisfactory. Also, it produces a livelier froth on the machine which is quicker breaking in the launders and pump boxes.

This frother offers a good possibility of improved metallurgy by raising concentrate grade and mineral recovery. DOWFROTH 250 demonstrates little or no collecting power, permitting independent regulation of frother and collector. Moreover, its water solubility permits the addition of DOWFROTH 250 as a water solution and assures return for reuse in mill water recovery systems.

DOWFROTH 250 requires practically no conditioning time and lends itself readily to stage addition. Adequate production facilities assure ready availability in both drum and tank car quantities.

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DOW

Manufacturers News

New Products • *FILL OUT THE COUPON FOR MORE INFORMATION* • Equipment

Payloader

The Frank G. Hough Co. has made available another four-wheel drive tractor-shovel. It has a 1 cu-yd bucket with 60 hp Diesel or 54 hp gasoline power optional. The use of large pneumatic tires, same size all around, and drive on all four wheels gives this new Payloader traction to work effectively in sand, mud and snow, and travel on pavements without injuring the surface. Rear wheel steer with power booster plus short wheel base makes it easy to operate and highly maneuverable. Four speeds in both directions insure fast travel speeds for transport and fast operating speeds on shuttle loading or similar operations. Bulldozer blade, crane hook, fork lift and snow plow attachments are available, quickly interchangeable with the bucket to further increase its multi-purpose usefulness. **Circle No. 1**

Belt Conveyor Idlers

The first basic improvement in years in belt conveyors idlers greasing design is announced by *Chain Belt Co.* The flexible sections are made in lengths which allow them to hug the inside of the inverted angle or channel base. Grease tube extension to the far side is accomplished by threading a flexible section through the base to the grease fitting at the near side. New, also, is the location of the grease fittings. They are placed so as to be more accessible, permitting safe servicing at any time and are protected during shipment by the idler base. **Circle No. 2**

Trailer

A new heavy-duty gooseneck-type trailer with tilting platform, for transporting heavy machinery with standard fifth wheel tractors, has just been announced by *La Crosse Trailer Corp.* Available in 14, 18 and 22 ton capacity, the new model GTTA trailer is of tandem axle design, with 96 in. width platform, which tilts into loading position by releasing simple lock at front of platform. Two double-acting hydraulic cylinders cushion load during tilting. After load is driven or winched into place, platform locks automatically in horizontal position for hauling. Trailer is ruggedly constructed with one-piece formed gooseneck and subframe, and is equipped with walking beams supporting the axles to provide maximum oscillation for equal load distribution. **Circle No. 3**

Respirator

The new respirator, called the Dustfoe No. 55, weighs only 2¾ oz, 25 pct less than previous models. Construction of the assembly allows for quick changing of filters and easy replacement of parts. The formable aluminum facepiece readily molded to all normal face contours and the replaceable tite-seal face cushion conform to all facial lines to assure a perfect seal without uncomfortable pressure. The preformed static web throw-away filter is shaped to fit the filter holder. The filter is made of charged resin-treated felt which creates a static electrical field and supplements the mechanical filtering action by trapping dust particles. *Mine Safety Appliances Co.* **Circle No. 4**

Backstop

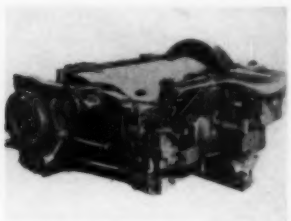
When electric motor current ceases, a vertical turbine water pump backspins unless means are provided to prevent motor reversal and avoid damage to pump bearings. A new method of preventing reversal of U. S. vertical hollowshaft motors has been developed by *U. S. Electrical Motors, Inc.* When the motor starts, centrifugal force throws the hardened steel balls out of engagement with the stationary member. When the motor stops, gravity pulls the balls down and one of them engages between the rotating member and the stationary ratchet, thus preventing reversal. **Circle No. 5**

Torque Converter

The Torcon Corp. announced recently the addition of the new model 17-K single-stage, three-element type unit to the standard line of Torcon torque converters. Combining a hydraulic torque converter and hydraulic coupling in a single unit, it provides automatic transmission for shovels and cranes, logging machinery, and material handling devices such as lift trucks and front end loaders. Major elements of the unit consist of the converter pump, converter turbine and reaction member. For continuous operating efficiency and simplicity of servicing, there are no close clearances in these elements — which are each single-piece aluminum castings. **Circle No. 6**

Diesel

Following thousands of service proven miles, *Cummins Engine Co., Inc.*, announces the availability of the new horizontal 200 hp model NHHB-600 Diesel. The NHHB-600 is a 6-cylinder, full Diesel, which produces 200 hp at 2100 rpm and with a compression ratio of 15.5 to 1. The weight of this unit is 2285 lb. Dimensions of the new engine are: Length 63 15/16 in.; width 55 1/4 in.; and height 22 3/4 in. **Circle No. 7**



Rotating Switch

A new rotating-type limit switch for application where reversing operation is to be coordinated with the number of revolutions of a motor shaft or driven equipment has been announced by the *General Electric Company's* Control Dept. The switch consists of two size 2 Switchettes each having one normally open and one normally closed circuit. Switch units are operated independently by cams rotated through a worm and gear reduction from a shaft extending outside the case. According to G-E engineers, simple adjustment of the new limit switch is accomplished by loosening two clamp screws, turning slotted adjusting pins (one for each cam) to the desired setting, and then retightening the clamping screws. **Circle No. 8**

Free Literature

(10) SHOCK ELIMINATION: A method of eliminating severe shock and vibration of hydraulically operated bucket loaders, power shovels, and other tracked or rubber tired equipment is described in a new bulletin No. 700 just published by *Greer Hydraulics*. Since hydraulic fluid is practically incompressible, the vehicle itself must absorb impacts caused when the loaded bucket is moved or stopped suddenly. By installing a Greer Hydro-pneumatic accumulator in the system, these shocks are absorbed fully. The accumulator also increases component life, relieves the equipment of structural fatigue and increases the production of the machine by adding to comfort of the operator.

(11) PROTECTIVE GLOVES: A new long-wearing all-purpose work glove with special coating that sheds moisture and resists chemicals better than rubber is described in a new bulletin of the *Mine Safety Appliances Co.* The vinyl-plastic coating stays flexible in extreme cold, does not become tacky at any high temperature to which workers' hands are normally exposed, and is especially effective in combating strong acid concentrations. The gloves will protect against concentrations of all acids, including sulphuric acid and oleum, and all caustics.

(12) STEEL FEEDERS: A new 20-page booklet covering *Pioneer Engineering Works* complete line of manganese steel feeders for the mines, cement mills and quarries has just been issued. The booklet explains the physical details of these feeders and gives operational data including capacities, hp and dimensions enabling the proper selection for every type of installation. These manganese steel feeders, called the Pioneer-Oro line, are designed and built for handling shock loads of heavy abrasive materials. The pans and all wearing parts are cast Supermang, a specially developed manganese steel alloy. Various types of drives are explained, as well as three systems for complete lubrication.

(13) CLASSIFICATION: This booklet issued by *The Dorr Co.* contains information on the DorrClone, a new cylindro-conical classification unit utilizing centrifugal force in place of gravity, and includes general description material, data on sizes and arrangements, applications, advantages and operating results. A key feature of DorrClone operation is the automatic underflow control device called the Vactrol. It has been found through experiments and practice that physical adjustments should be made to the cyclone in applications where there is a considerable change in feed solids.

(14) WIRES AND CABLES: A new 52-page engineering catalog on its line of electrical wires and cables for the coal mining industry has been published by the electrical wire and cable department of *U. S. Rubber Co.* The booklet includes complete performance and specification data on insulation and jacket compounds, portable cords and cables with a voltage rating up to 5000 v. Also included are complete data on shielded portable cables, welding cables, bore hole cables, mine power cables and miscellaneous mine equipment such as blasting wires, miner's lamp cord and mine telephone cable.

(15) SAFETY CONNECTING LINKS: Wedglok safety connecting links are used to connect either chain or wire rope in all kinds of assemblies. You can get links in any one of five different designs, such as standard for attaching hooks, pear shaped for attaching top rings, and universal links which are specially designed to cover a wide variety of uses. These links are used in material handling operations, in leading steel mills, foundries, industrial plants, and construction jobs. They are used in strip mines, gravel plants and on all types of dragline bucket operations. *Interstate Drop Forge Co.*

(16) COAL EQUIPMENT: A new 16-page booklet released by *Allis-Chalmers* on their vibrating screens, centrifugal pumps, motors, motor starters, V-belt drive equipment, car shaker and power and electrical equipment. Construction features and ratings of pumps and motors are summarized and a table shows the scope of the company's line of motor starters. In each instance, the guide gives numbers of bulletins which provide a more complete description of the various equipment covered.

(17) SHUTTLE CARS: The chassis consists of a heavy zee bar on each side of the car, connected by steel tubes. These tubes are welded to the heavy zees in such a manner as to make a rugged structure that will withstand any stresses to which the chassis is subjected. The car is driven by two series wound motors, one driving the two wheels at the discharge end and one driving the two wheels at the loading end. The conveyor chains and flights are driven by a hydraulic motor with variable speed control to regulate rate of discharge. The cable reel is driven by a hydraulic motor, the cable reel operates automatically without any attention from the operator. *Jeffrey Mfg. Co.*

(18) BUCKET CHAINS: This bulletin deals with Tisco manganese steel drag line bucket chains offered by the *Taylor-Wharton Iron & Steel Co.* These bucket chains are made of Hadfields manganese steel, containing 12½ pct manganese, upward of 1 pct carbon. The Tisco chain is cast in interconnected molds, each chain being poured from the same heat of steel to insure uniformity of analysis and structure. Connecting links are designed for easy assembly. The key serves as a reinforcing stud and the rivet is not in shear. They can readily be installed by hand.

(19) DIESEL ENGINES: Design and operation features of the Nordberg one, two and recently introduced three cylinder type 4FS Diesel engines are illustrated and described in a new 12-page bulletin published by *Nordberg Mfg. Co.* Built for continuous duty or standby service, these engines have a 4½ in. bore and 5¼ in. stroke and are of the four-cycle, vertical mechanical injection type. The units range from 10 to 45 hp within an operating speed range of 1200 to 1800 rpm.

Mining Engineering **March**
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Lower initial cost, lower operating cost, and sharper separations—these are the advantages of selecting the right diameter, length, depth, tank design, and details for **your particular material and operating conditions.** Only the WEMCO S-H Classifier offers such a wide selection—and WEMCO's metallurgical staff is ready to aid you in applying the correct unit.

The following table shows a partial breakdown of more than 2,000 standard WEMCO Classifier models:

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1 SPIRAL DIAMETERS		A right diameter for every job, giving better balance between sand conveyance and pool area requirements.
2 SPIRAL PITCH		"S-H" (Special Helix) advanced pitch design on all models, giving up to 100% greater raking capacity compared to other designs.
3 SPIRAL LENGTH		Length furnished to fit exact needs of each job—correct lengths for closed circuiting—adequate drainage deck length for dewatering.
4 TANK STYLE		Tank styles to fit the characteristics of the material and the separation to be made—up to 25% greater "effective" pool area than other units.
5 POOL DEPTH		Generally Series 90 is used for 48 mesh and coarser separations, Series 125 for 48 mesh to 150, and Series 150 for 100 to 325 mesh separation.
6 ASSEMBLIES		Duplex units give nearly double the capacity of simplex units, but require less floor space than two separate units.
7 LIFTING DEVICE		Smooth, powerful hydraulic action is usually preferred to the older style screw lift.

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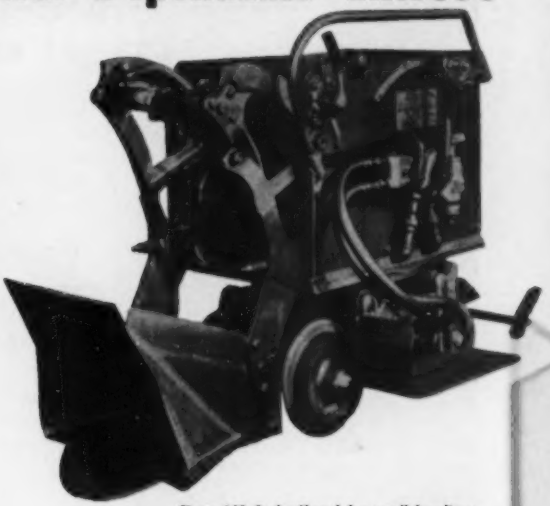
Speed Up Loading with Dependable EIMCOS



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Eimcos do a better job in loading because they're actually top quality machines. They're powerful, efficient and rugged. The use of alloy steel castings and anti-friction bearings make them the most dependable loader on the market. Exclusive design features make them safe and free from troublesome gadgets—easy to operate.

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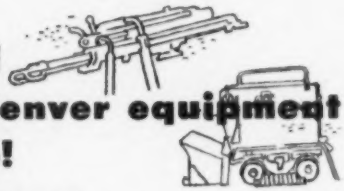

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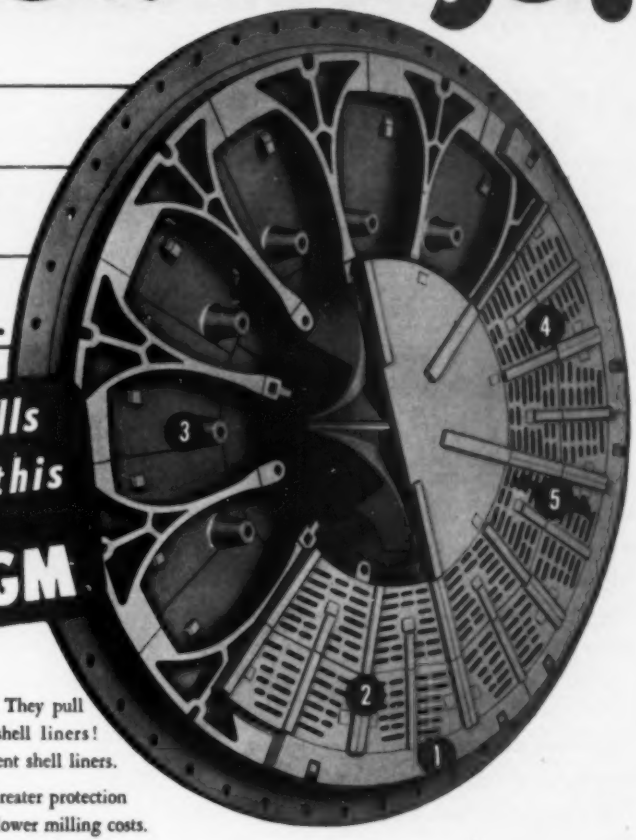

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NI-HARD widely used ...where abrasion is severe

ONE OF THE HARDEST commercial products of the iron foundry is a *nickel-chromium white cast iron* trade-named "NI-HARD*".

HARDNESS RANGE

Its Brinell hardness ranges from 550 to 650 when sand cast, and 600 to 725 when chill cast.

This extreme hardness . . . at the moderate price of NI-HARD . . . provides abrasion resistance at lowest ultimate cost.

STRUCTURE

Unlike other cast irons . . . NI-HARD comprises a multitude of hard carbides firmly embedded in a matrix which is as hard as fully hardened steel. NI-HARD is unmachinable except in special cases. Finishing requirements are met by grinding or the use of cast-in-place machinable inserts.

PERFORMANCE

NI-HARD develops abrasion resistance two to three times better than that of unalloyed white iron. Therefore NI-HARD can economically replace white iron.

NI-HARD lasts one and one-half to two times as long as austenitic manganese or carbon steels, under conditions involving only moderate impact. Chill cast parts are stronger and more abrasion-resistant than comparable sand castings.

STRESS RELIEF

The strength and toughness of NI-HARD castings are increased fifty to eighty per cent, without loss in hardness or abrasion resistance, by a stress relieving treatment at 400-450 F. User experience has demonstrated the merit of specifying this treatment.

APPLICATIONS

Applications include: grinding balls, ball and rod mill liners, slurry pump parts, flotation impellers, piping, scoop lips, classifier shoes, roll heads, pulverizer rings, chutes and hopper liners, muller tires, plows, pug mill knives, clay augers. Also coke grizzly discs, crusher rolls, coal mine pumps, dredge pump impellers, liners, rings and sleeves. Also industrial food grinding burrs and attrition mill plates.

AVAILABILITY

At the present time, the bulk of the nickel produced is being diverted to defense. Through application to the appropriate authorities, nickel is obtainable for the production of NI-HARD for many end uses in defense and defense supporting industries. There are authorized foundries, from coast to coast, equipped to quote you on NI-HARD castings in all common forms and shapes.

INFORMATION AVAILABLE

Write for the booklets, "Engineering Properties and Applications of NI-HARD", and, "Buyers Guide for NI-HARD Castings."

*Reg. U. S. Pat. Off.

EMBLEM OF INCO SERVICE

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

Manganese ore from the Black Rock, Kramer, and Black Jack mines in Tooele and Jaub Counties, Utah, are of acceptable grade, the U. S. Bureau of Mines has announced.

Payment of a premium price of 25½¢ per lb to the Copper Range Co., of Michigan for the output of its White Pine mine, has been agreed to by the DMPA. The contract will be limited to $28.6 million worth of copper. The agreement follows an RFC loan grant of $57 million to bring the White Pine property into production within three years.

Increases of 5 to 6 cents over the Lake copper ceilings have been granted to the Calumet & Helca Consolidated Copper Co., for operation of "high cost" deposits, according to Jess Larson, DMPA administrator. Mr. Larson expects the move to add about one million lb of copper per month to the supply. The agreement can be cancelled on 60 days notice and is terminated automatically in the event copper ceilings are removed.

Reynolds Metals Co. starts bauxite shipments from its rich Jamaica, B. W. I., deposits in September or October to the Hurricane Creek plant. The Jamaican imports, intended to supplement Arkansas bauxite, will reach an annual rate of about 500,000 tons. The new aluminum reduction plant under construction at Corpus Christi, Texas, will make use of the Jamaican bauxite.

Largest of the certificates of necessity for accelerated tax amortization of new or expanded defense facilities to gain approval went to the Reserve Mining Co., Beaver Bay, Minn. It involved the company's proposed investment of $105.8 million for the production of iron ore.

Extensive new discoveries of monazite, a mineral containing thorium, has been made by John Mertie of the U. S. Geological Survey. No disclosure as to the exact extent of the discovery is available because of the importance of thorium as atomic energy material. However, it is known that the deposits extend about 700 miles through the Southwestern states.

Tremendous pressures are being applied to bleed the stockpile of scarce metals, according to John D. Small, Munitions Board chairman. Mr. Small said that all efforts to deplete the stockpile will be resisted until every other avenue of supply has been investigated. He noted that the shortage of steel, copper and aluminum will probably continue through June.

Drainage tunnel systems in the Leadville, Colo. district have been completed and further exploration of the district that has already produced $460 million in metals can be done. Shaft measurements and reports indicate that the water has been lowered to the tunnel level in the Carbonate Hill basin; Fryer Hill basin workings were drained earlier.

armed guards

proved very effective in increasing ore production in the old days. "Feather-bedding" was totally absent among Apache prisoners condemned to the ore mines of Mexico.

The Traylor TC Gyratory, with its curved concaves and bell head, is a perfect example of advanced crusher design. Bulletin 128 gives complete details.

THE PRODUCTION of ore has passed through many interesting and adventurous phases. For the past 50 years, Traylor has contributed to its growth and development. As the need for more efficient, more productive machinery increased, Traylor kept pace with constantly improved, more dependable equipment. By working with the mining industry for half a century, Traylor knows its problems . . . builds equipment "Traylored" to its needs.

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JEFFREY BELT CONVEYOR

● An economical means of advancing transportation is provided with this Jeffrey 64-A Sectional Belt Conveyor with its intermediate sections.

It is frequently used where ore and other material must be moved continuously over long distances . . . has been operated in lengths exceeding 3,000 feet.

The Jeffrey 64-A Belt Conveyor is available in three belt widths—30, 36 and 42 inches. Range of speeds vary from 175 to 500 feet per minute—capacities up to 18 tons per minute, depending upon speed and size of belt.

While built in standard units for quick, low cost extension of the haulage system, each conveyor is engineered to meet your conditions. For fast, economical transportation—above or below ground—consult a Jeffrey Engineer.

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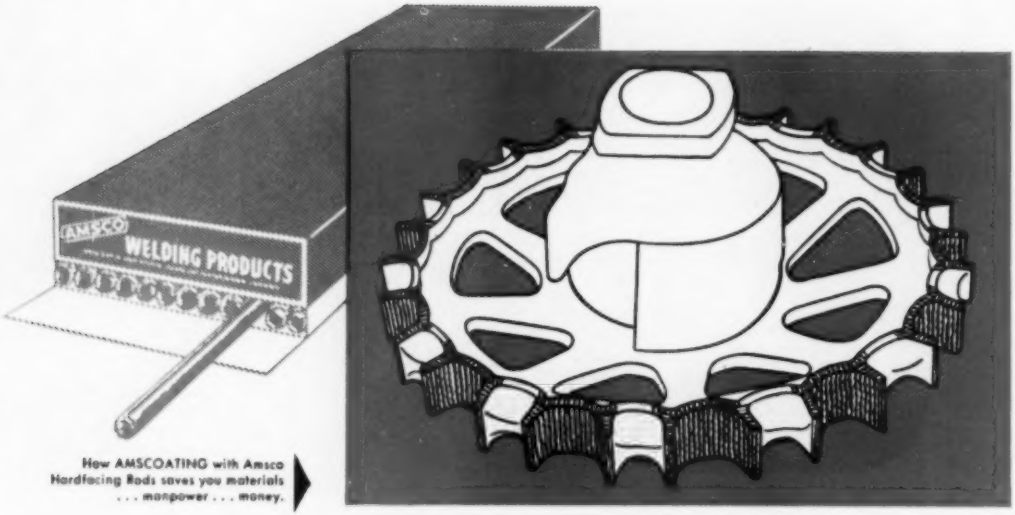

How AMSCOATING with Amsco Hardfacing Rods saves you materials . . . manpower . . . money.

These teeth take bites out of repair costs!

AMSCOATING can make old Tractor Drive Sprockets last 3 times longer than new.

Here's another way to beat the high cost of replacements, down-time and maintenance . . . AMSCOATING the teeth on Tractor Drive Sprockets. Time after time service records have shown that AMSCOATING of sprockets results in these dollar-saving advantages:

1. ***On an average — 3 times the service life of a new, unhardfaced sprocket.***
2. ***2 out of 3 sprocket changes completely eliminated . . . far less down-time, less maintenance . . . more equipment out of the shop and on the job.***

To top it off, sprocket replacements can practically be eliminated. An AMSCOATED sprocket can be reclaimed over and over again . . . *at about half the cost of a new one!*

Get all the facts on how to make these big savings . . . write today for instruction sheet on AMSCOATING sprockets, and the name of your nearest AMSCO Distributor.

AMSCOATING...stands for control of wear by Hardfacing...

Hardfacing rods—and recommendations for their use—are as sound as the manufacturer who makes them. AMSCO has been fighting wear for a half-century—first with Manganese Steel, and later with AMSCO Hardfacing Products.

If you have a problem of wear caused by impact, abrasion, heat or corrosion . . .

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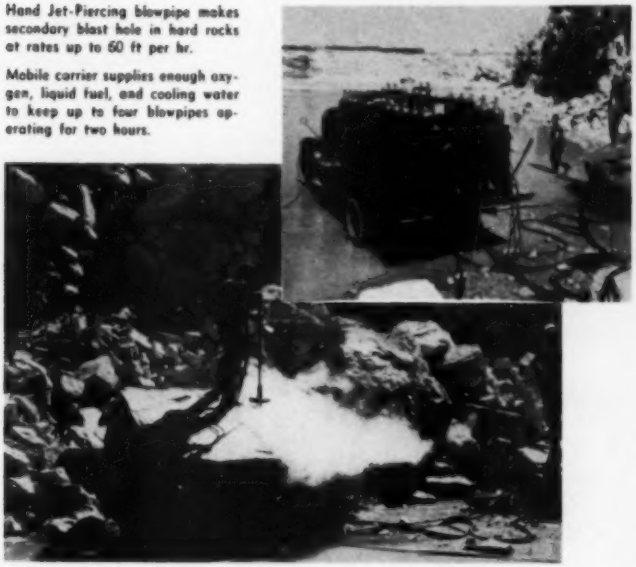
Amsco Welding Products distributed in Canada by Canadian Liquid Air Co., Ltd.

Hand Jet-Piercing blowpipe makes secondary blast hole in hard rocks at rates up to 60 ft per hr.

Mobile carrier supplies enough oxygen, liquid fuel, and cooling water to keep up to four blowpipes operating for two hours.



New Jet-Piercing Development Speeds Quarry Operations

Secondary blast holes are now being pierced in quartzite and sandstone quarries, using multiple hand-operated Jet-Piercing units supplied with oxygen, fuel and water from a compact mobile carrier. A unit of this type, operating in a Canadian quarry, enabled four Jet-Piercing blowpipes to pierce blast holes for two hours without shutdown.

The piercing blowpipes used were developed by Linde Air Products Co., a division of Union Carbide and Carbon Corp. The mobile supply carrier was constructed by the quarry from specifications supplied by Linde and modified by the quarry operators for their particular use.

Employment of the new self-contained unit, which allows up to four operators to pierce approximately 450 ft of blast hole without shutdown, accelerates rock removal operations. Normally, only two blowpipes are in operation at one time. Large boulders or ledges, which sometimes remain after primary blasting to hamper loading operations, can, in the case of certain hard rock, be prepared for reduction more rapidly with these new units than by any other means.

A special feature of the new manual Jet-Piercing unit is the mobile gas and fluid supply carrier. It consists of a standard truck chassis fitted out with four banks of large standard-capacity oxygen cylinders, a steel-holding tank for fuel oil, and a gasoline-driven air compressor and tank for pressurizing the fuel oil tank. Cooling water for the blowpipes is supplied to the unit from a hose extension to the quarry supply line. Feeder lines from the oxygen cylinder manifolds and the oil and water tanks terminate in four sets of take-off stations on the control panel at the rear of the truck. As many as four hand Jet-Piercing blowpipes can be connected to the take-off panel by high-pressure rubber hoses.

The hand Jet-Piercing blowpipe used with the mobile supply carrier is a highly portable device which can be manipulated comfortably by one operator. The blowpipe can pierce blast holes at rates up to 60 ft per hour, depending on the properties of the rock. Blast holes can be pierced at any angle from the vertical to the horizontal, making it necessary for the operator to drill from the top of the boulder. Hole diameter is held to approximately that which allows the use of a standard 1⅛-in. diameter dynamite.

The manual Jet-Piercing blowpipe has these important design features: a burner from which the combustion gases emerge at supersonic speeds (in excess of 6000 ft per sec); a hole-sizing shell which controls the minimum size of the hole and protects the copper burner; and a sliding shield which protects the operator against the steam and spallings ejected from the hole during the piercing.

The Jet piercing process has been applied successfully in United States and Canadian quarries within the last several years as an advanced method of "drilling" blast holes.

Newfoundland to Start Intensive Exploration

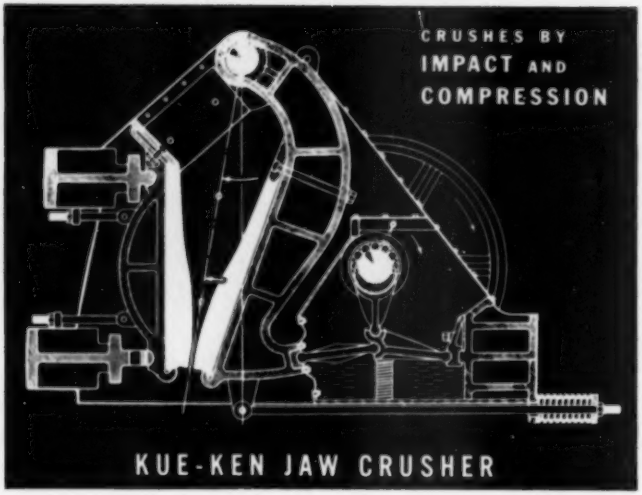
Financiers in the United States and Canada have joined with the Newfoundland government to launch an extensive search for the potential resources of that country.

The combination of government and business is called the Newfoundland and Labrador Corp. and has an authorized capital of $10 million. Exploration of Newfoundland's 110,000-sq-mile triangle will start in the spring in an effort to learn the extent of the mineral wealth buried there.

Mining experts have hopes that Labrador may prove to be the richest mineral region in Canada. When the minerals are found, there will be no shortage of power for mining or smelting. Waterpower in rivers and waterfalls will develop more electricity than will be required. The vital railroad facilities that would be needed in future development are under construction.

Smelter Gases Produce Liquid Sulphur Dioxide

A Canadian development will help meet the current shortage of sulphur and utilize a byproduct of smelting that often goes to waste. The International Nickel Co., at Copper Cliff, Ont., has developed a flash-smelting process for ore concentrates. By using oxygen instead of air, the sulphur dioxide content of the gas produced can be increased from 5



MAYBE IT'S A JAW CRUSHER

It could be that a jaw crusher is the ideal type of crusher for your reduction job. If so, most people that know say the Kue-Ken is the best of all jaws. But it's entirely possible that another type of crusher or grinder will do the job better and at less cost. If that's the case, we'll tell you without hesitation, because we make a large and varied line and it's to our advantage to see that you get the machine that's best for your job. Our job analysis service won't cost you anything and it may save you a lot. Write us.

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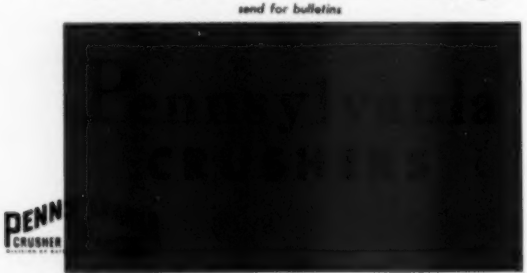

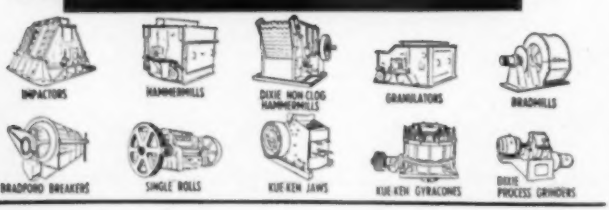

pct to 75 pct. Using this rich gas, a process has been developed by Canadian Industries Ltd., at its Copper Cliff plant, to make liquid sulphur dioxide. The Abitibi Power and Paper Co. has proved that this product can economically take the place of sulphur as a raw material for making sulphite paper pulp. A commercial plant is being erected, to be in operation this year, which will have an annual capacity of 90,000 tons of liquid sulphur dioxide a year, equivalent to 45,000 tons of elemental sulphur. Product of the new plant is believed to equal more than the present production of liquid sulphur dioxide in the entire free world. It will practically all be used in the Canadian sulphite paper mills.

New Operations Ease Sulphur Shortage

The sulphur shortage which burst on an unsuspecting world some two years ago has been met by Canadian industry, and this year sees new supplies of domestic output.

In 1952, at least 85,000 tons of annual sulphur capacity will be brought into being in Canada. The first of the new sulphur plants started operation when the Jumping Pound plant of Shell Oil Co. came "on stream," pouring great golden cakes of elemental sulphur won from the "sour" gas of the Jumping Pound field, 20 miles northwest of Calgary, Alta. Capacity is 30 long tons per day, or 10,000 tons per year.

Before the year ends, production will have begun at the Royalite Oil Company's plant of similar capacity, situated in Alberta's Turner Valley field. That makes a total of 20,000 tons of annual production from gas piped from reservoirs below the Western Canada plains.

In the next few months, Nichols Chemical Co. will complete a $2.5 million expansion project to double the acid production of its Valleyfield works. Canadian pyrite supplies the raw material for this and the company's other two acid plants.

Noranda's proposed plant, which would roast 300 tons of pyrite daily for an annual output of about 18,000 tons of elemental sulphur, plus the equivalent of 35,000 tons of sulphur as sulphuric acid.

Totalling up the above, 85,000 tons of elemental sulphur, or equivalent, will be added this year.

Titanium in Nevada

Titanium, the new metal half as light as steel and about twice as strong, is now in limited production at Henderson, Nev., in a plant built during World War II to turn out magnesium.

A portion of the one-time magnesium plant is being rehabilitated to expand production of this noncorrosive, heat-resistant metal to

3600 tons annually—a goal expected to be reached before the end of the year. Initial production was achieved last fall, only four months after the award of engineering and construction contracts to the Ferguson Co. The new metal is needed in the defense program, where it is used for atomic energy, jet aircraft and other essential applications. The Henderson plant is operated by Titanium Metals Corp. of America, a subsidiary of National Lead and Allegheny Ludlum Steel Corp.

Plans For New Taconite Development

A large-scale development in Minnesota for producing up to 10,500,000 tons annually of iron ore pellets from taconite is under consideration by Erie Mining Co. Erie is owned by Bethlehem Steel Corp., The Youngstown Sheet and Tube Co., Interlake Iron Corp. and Pickands Mather & Co.

An application for a certificate of necessity covering the entire project on a basis of 10,500,000 tons, estimated to cost about $300 million has been filed with the Defense Production Administration. Final decision to undertake the development depends upon the action taken by the Government on the Company's application and the solution of certain tax problems, according to Mr. Elton Hoyt II, president of Erie and senior partner of Pickands Mather & Co.

The program calls for constructing a plant on the Mesabi Range northeast of Aurora, Minn., to produce the concentrated iron ore pellets from taconite, the low-grade formation which abounds in Minnesota. It also calls for building a new community east of Aurora for the plant employees and their families, a large power plant, and a railroad to carry the pellets from the plant to docks and loading facilities at Two Islands, Minn., on the north shore of Lake Superior.

If the project is undertaken, the initial unit would have an annual capacity of at least 5,000,000 tons, Mr. Hoyt said, which would probably take about four or five years to attain. According to present plans construction of additional units would continue until production is at the rate of 10,500,000 tons per year. This procedure would assure maximum production at the earliest possible date.

Erie Mining Co. for several years has operated a preliminary taconite plant at Aurora for experimental purposes, which last year produced and shipped about 70,000 tons. This is the only plant in existence which has produced pellets in large quantities from Lake Superior taconite.

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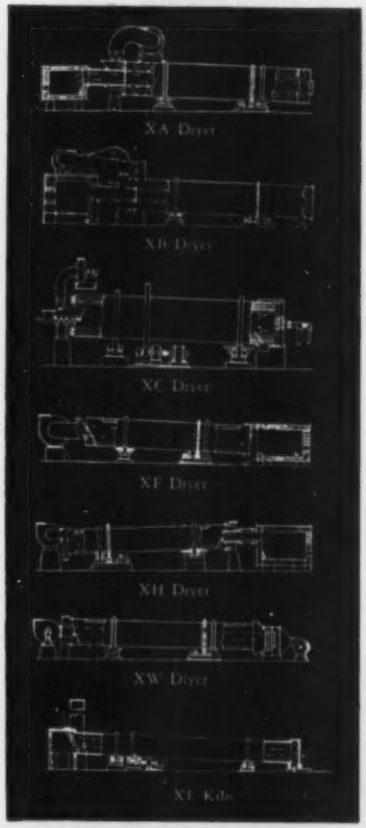
DRYING

YOUR PROBLEM?

RUGGLES-COLES DRYERS



Ruggles-Coles Rotary Dryers, built by Hardinge, are available in the following styles:

CLASS XA — Double shell, semi-direct heat. High efficiency. For materials that can be dried in direct contact with combustion gases and heated above 212°.

CLASS XB — For materials that must be dried by indirect heat but can be heated above 212°. A double shell dryer with low dust loss.

CLASS XC — Steam tube dryer for materials that must be dried by indirect heat and at low temperatures, such as chemicals, grains, food products.

CLASS XF — For direct heat drying at temperatures above 212°. A single shell counter-flow dryer that does not discharge the material through the furnace.

CLASS XH — For direct heat drying at temperatures above 212°. A single shell parallel flow dryer designed to handle sticky materials.

CLASS XW — For material that can be dried by hot air at temperatures below 212°. Dries ammonium nitrate; potassium chloride, etc.

KILNS — Rotary type for drying, calcining, roasting or oxidizing at temperatures above the range of ordinary dryers. Refractory lined.

COOLERS — Air, water spray or submerged rotary type. Each designed for the efficient cooling of materials from kilns or roasters.

Bulletin 16-D-1 describes entire Ruggles-Coles line.

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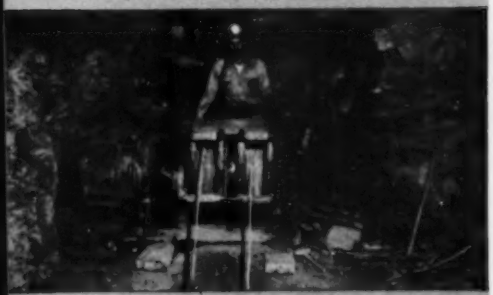
Joy AAF-211 double-drum slusher, powered by 15 H.P. electric motor, in operation in a western copper mine, scraping into a grizzly.

Joy E-111 Turbinair single-drum hoist on a timber setting operation in a large western mine.

Joy HL-3 Shovel Loader mucking out a drift round in a western copper mine.

Built with JOY'S intimate knowledge of mining problems... they're scraping millions of tons of muck every year

Turbinair driven 5 H.P. S-211 slusher scraping ore in a stope in a western fluorspar mine.



Electric driven CFA-211 double-drum slusher in operation in a Tennessee zinc mine.

25 H.P. B-312 electric slusher scraping ore in a large western Canadian mine.

Ruggedly built to stay underground longer, JOY Slushers stand up under the heaviest loads and toughest conditions to give you that high efficiency found only in machines which are *designed* for the job and *proved* in the mine. Check these features: they assure more tonnage scraped per shift and longer life with less maintenance.

1. Anti-backlash brakes
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● There's a JOY Slusher in a size and type for every scraping job. You need only to consult a Joy Engineer to get your answer. He can recommend the best loading equipment for your needs because Joy manufactures the only *complete* line of rock loading equipment for mines . . . slushers, track-mounted shovel loaders, and continuous-type trackless loaders.

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FOLLOWING the major explosion disaster in the Orient No. 2 mine of the Chicago, Wilmington and Franklin Coal Co., West Frankfort, Ill., on December 21, 1951, resulting in the death of 119 miners, an investigation was made by the U. S. Bureau of Mines under the direction of J. J. Forbes, Bureau director. The subsequent report concluded that the disaster was caused by the emergence of a large body of explosive gas from an abandoned area and simultaneous short circuiting of the ventilating current; the gas was ignited by an electric arc from nonpermissible electrical equipment and the flame was propagated throughout a large part of the affected area by coal dust and possibly by gas from other worked-out and abandoned areas.

An immediate repercussion of the disaster was the opening of hearings by the Senate Labor and Public Welfare Subcommittee on the Neely Mine Inspection Bill (S. 1310, introduced April 11, 1951) which would give federal coal-mine inspectors power to enforce safety recommendations by ordering the withdrawal of all employees where conditions constitute an imminent danger to their safety. Two days of testimony, January 24 and 25, disclosed that coal operators were not in agreement on the bill, although John L. Lewis, speaking for the UMWA, and J. J. Forbes, for the Bureau of Mines, were advocates.

Robert E. Lee Hall, counsel for the National Coal Association, declared the bill unconstitutional, in contravention with states' rights, and would not promote safety in the mines. He warned that a shift in the responsibility of enforcing mine safety laws from state governments to the federal government could bring about "serious curtailment" of the activities of state mining departments. Walter R. Thurmond, representative of the Southern Coal Producers' Association, also opposed the bill on the ground that it was an infringement on rights reserved to the states and also that it could contribute nothing constructive to the work already being done by the industry, state mining departments, and the Bureau to eliminate hazards.

Favoring the Neely bill in principle, but not in its present form, was Ed D. Schorr, who spoke for the Ohio Coal Producers' Association and other operators who produce a total of approximately 103 million tons of coal. He stated that the operators which he represented did not think that federal enforcement of mine safety laws would injuriously affect mine safety. He urged that Congress itself should enact appropriate mine safety laws rather than delegate the authority to any federal administrative agency and also that adequate appeals from and reviews of orders to close mines be provided. Mr. Schorr introduced amendments of S. 1310 and recommended that they and the existing Federal Mine Safety Code be enacted into law.

J. J. Forbes cited the need for uniformity in safety laws throughout the coal-producing states. He mentioned that "the underlying unsafe conditions and practices that caused the Orient disaster also exist in many other coal mines in this country and constitute a real threat to the economic life and defense of our nation." In tracing the history of the Bureau of Mines, he said that as a result of a series of disastrous mine explosions during the years 1906 to 1910 public sentiment was aroused and Congress established the Bureau of Mines in 1910, primarily to promote health and safety work through educational, investigative, and research work. In 1941 the Federal Coal Mine Inspection and Investigation Act was passed which made it possible to apply safety standards on a nation-wide basis. This act gave authority to federal inspectors to enter coal mines, make inspections, and submit recommendations for preventing accidents and improving health conditions, but contained no provision for mandatory compliance with the federal recommendations. During the period December 1, 1941 to May 31, 1946, when federal inspections were based on the Bureau of Mines Safety Standards, only 25 pct of the federal recommendations were complied with. During the period June 1, 1946 to March 25, 1947 (immediately preceding the Centralia disaster) the United States Government had technical control of the nation's bituminous-coal mines. Compliance in government-controlled mines was 28 pct, and 16 pct in mines not so controlled.

During the period March 26, 1947 to June 30, 1947 (immediately following the Centralia disaster), when authority was granted federal inspectors to close hazardous government-controlled mines, compliance in such mines was 48 pct, and 22 pct for mines not so controlled. Following the Centralia disaster, March 25, 1947, in which 111 men were killed, the Congress passed Public Law 328 requiring notification by the mine operator and the state mining departments of the degree of compliance with such recommendations. During the year covered by Public Law 328, 59 pct of the federal inspectors' recommendations were not complied with. According to Mr. Forbes, state officials have the power to enforce compliance with state laws but he concludes that either the laws are inadequate or are not properly enforced.

At the request of UMWA, Bureau of Mines personnel prepared revisions of the mining laws of several coal-producing states in line with the Federal Mine Safety Code. These have been adopted by Alabama, Colorado, Maryland, Montana, Tennessee, and Wyoming. Based on these facts and others stated at the hearings, Mr. Forbes favored federal enforcement authority of coal mine safety.

John L. Lewis refuted the generally accepted belief that the Mine Safety Committee could withdraw the miners from an unsafe area when the federal or state inspector finds a dangerous condition. He said that under the Taft-Hartley Law this would be a breach of contract. In rebuttal to this declaration by Mr. Lewis, Senator Taft replied that Lewis was wrong in believing that the Taft-Hartley Act prevents the miners from walking out of unsafe mines in "good faith" and said that the operators would have the chance of a "snowball in hell" of suing for breach of contract under such conditions.

Informed sources on coal mine safety point out that the problem has its economic aspects. Insurance rates are predicated on accident statistics, making safe working conditions important for economic operation. An inspector with authority to withdraw the men from a mine would have to weigh carefully the degree of hazard against the loss in wages to the miner and the loss of coal production to the national economy. The Mine Safety Committee now faces the same problem in balancing the accident probability against lost wages.

Although nearly everyone agrees that the Neely bill cannot eliminate accidents or disasters, there are experienced people in mine inspection work who say that the bill would be detrimental to the steadily improving safety record. The safety record achieved by the coal operators and interested agencies shows a steady decline from 3.60 fatalities per million tons produced in 1926 to 1.29 in 1951 or 2.002 fatals per million man-hr of exposure in 1926 to 1.08 in 1951. They point out that safety cannot be legislated; it is a process of education and constant vigilance. Laws are necessarily and inherently a compromise, and obviously safety cannot be compromised. It is pointed out that the Bureau stamp of permissibility on mining equipment does not preclude the possibility that the equipment might not

be maintained in permissible condition. A mine may be safe when the inspectors are present but the condition can change in a matter of hours. For these and other reasons, some sources maintain that police power granted to federal inspectors might create a false sense of security and defeat the many years of achievement in safety education.

In some states a good job is being done in enforcing state laws, resulting in maintenance of good safety standards. It is pointed out that the Neely bill might interfere with this good work being done and cause the states to withdraw from the work of coal mine inspection. An argument favoring state inspection and state codes is that a practice permissible in one type of mine or mining conditions may not be safe under other mining conditions.

From the above it can be seen that Congress is faced with a highly controversial subject but it is believed that some safety measure will be reported by the subcommittee.

TWO for one will be the ratio of aluminum production in 1953 to current production if present plans reach fruition. Of course the defense effort is responsible for this enormous expansion program. During World War II aluminum capacity reached a peak of about 1.8 billion lb which compares with present production of about 1.7 million lb. The casual observer will point out that the aluminum industry will have excess capacity with over 3 billion lb in 1953 if defense requirements slacken.

The industry, however, is thinking in terms of a permanently expanded primary metal capacity, and, taking a page out of Inco's book, research and development of new uses is being pushed. This research is getting a lot of help from the outside—copper consumers in particular. When NPA pointed out that although copper and aluminum were in tight supply, aluminum would be easier in the future, some manufacturers took the cue. General Electric has freed approximately 10 million lb of copper by substitution of aluminum and other metals.

Aluminum is handicapped in the electrical field as its conductivity is only 64.9 pct of copper. Intensive research is overcoming this obstacle, an example of which is the new aluminum-base lamp bulb. G. E. has been successful in the development of aluminum substitution in some components of transformers, washing machine motors, and turbine-generator field windings. In this latter application a new alloy has been developed, Cond-Al, which is expected to save 200,000 lb of copper this year in the turbine plant.

Biggest aluminum-consuming industry has been construction, with transportation important and growing and power transmission substantial. Agricultural consumption is new and growing as a customer. In this field it is used for roofing and irrigation pipe.

The light metal industry hopes to at least hold its ground on the price advantage it presently enjoys; aluminum pig sells for 18¢ per lb. Because it takes 10 kw hr to produce one pound of primary aluminum, the industry gravitates toward cheap power sites. Of the six new plants, two will be in the Northwest using hydroelectric power; four will be in the South of which three will generate electric power using natural gas as a fuel and one lignite. This will be the first time lignite is used as a source of power for aluminum.

Other methods are being studied to reduce the cost of producing primary aluminum. The industry is working with the Atomic Energy Commission to develop electric power with atomic energy. In addition chemical methods of reducing alumina to aluminum are being studied.

At the present time there are three companies in the field of aluminum, Aluminum Company of America, Reynolds Metals, and Kaiser Aluminum & Chemical. Alcoa has two new plants under construction for completion in 1952, one at Wenatchee, Wash., and the other at Rockdale, Texas. New and expanded capacity will bring production of that company from 852.5 million lb to about 1.26 billion lb in 1953. Reynolds is planning new capacity at Corpus Christi, Texas, and in Arkansas. Present production is 496 million lb and will be expanded to 813 million in 1953. Kaiser has a new plant planned at Chalmette, La., which, with expansion of the Mead, Wash., plant, will increase capacity from 350 million to 790 million in 1953. Anaconda Copper Mining Co. and Harvey Machine Co. are entering the primary aluminum field and should have their plant at Kalispell, Mont., completed by 1953. Capacity will be 144 million lb.

FREE-WORLD nickel production in 1951 is up 10 pct over 1950 according to the review of the nickel industry recently released by International Nickel. Estimated at 295 million lb of which 275 million lb comes from Canada, the free-world production is thought by J. F. Thompson, president of Inco, to be several times greater than production behind the iron curtain.

About 30 pct more nickel is expected by 1954 because of expansion of existing facilities at Inco's Creighton mine and Falconbridge Nickel Mines and of the Nicaro nickel plant and Sherritt-Gordon's Lynn Lake discovery in Manitoba.

Nickel has been in short supply during the year because of peak military demands and stockpiling causing the voluntary distribution plan in force during the first half of 1951 to give way to NPA allocation at midyear.

WORLD gold production, increasing by about 3½ pct annually since 1947, reversed its trend in 1951. Production in 1951 was about 24 million oz as compared with 24.3 million oz in 1950. This fact and other highlights of the 1951 gold situation were reviewed by M. A. Kriz of the staff of the New York Federal Reserve Bank, who customarily prepares a memorandum on this subject each year. This reversal in trend is interesting particularly because it indicates that the 1949 currency devaluations, by countries which produce 80 pct of the world's gold, have had a negligible effect as a production stimulus.

Premiums paid on gold declined substantially. Where dollars were involved, prices fell from $44 per oz a year ago to $38.75 in December 1951. Black market prices abroad ranged from $42 in Paris and Hong Kong to $45 in Alexandria and $60 in Bombay.

The outflow of gold from the United States, which began with the devaluations in 1949, reversed itself spectacularly by the third quarter of 1951. Net sales, which had totaled $880 million in the first quarter of the year declined to $57 million in the second quarter, and by the third quarter had given way to net acquisitions of $290 million. The most striking reversal occurred in the transactions between this country and the sterling area; net sales in the first quarter totaling $400 million were replaced with net receipts of $340 million in the third quarter.

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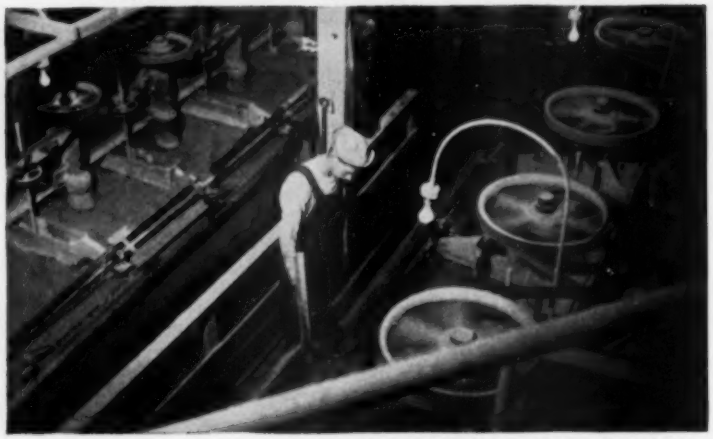

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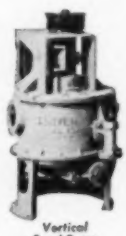

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MINING ENGINEERING

EDITORIAL

NO SECRET ABOUT SAFETY

"IT is decreed by Divine Providence that those who know what they ought to do and then take care to do it properly, for the most part meet with good fortune in all they undertake; on the other hand, misfortune overtakes the indolent and those who are careless in their work." These are the words of Georgius Agricola to those who are engaged in mining, taken from *De Re Metallica*, Book II, second paragraph.

Safety in mining is just that simple, when considered on the basis of the individual. The words imply a knowledge of what is to be done and how to do it. Armed with this knowledge it is only necessary to proceed with diligence and care to achieve the desired end without personal or financial mishap. Obviously the complication arises in the large company operating numerous mines and the consequent ramifications of company structure embodying management, supervision, engineering, and labor. Agricola stresses later on the importance of supervision and direction by the owners of the mine. We know that this principle is recognized today by the frequency with which the statement is repeated "a safety program must stem from management." People charged with responsibility for safety and management also stress the importance of securing the cooperation of the employees in formulating and carrying out safe working practices. Safety committees consisting of employees, supervisors, and representatives of management are recognized as the most practical method of achieving safety consciousness in all levels of mining company organization.

These facts being accepted by all those engaged in mining, why then the repetition of safety creed and slogans? The answer is that even though the number of accidents and severity is steadily diminishing, we can do better by education for the emergency when the supervisor is not present.

We have quoted to you a 400-year old authority and now we wish to quote, from the present, a letter to all anthracite mine inspectors, officials, and miners from Joseph J. Walsh, deputy secretary of mines, Commonwealth of Pennsylvania. After Mr. Walsh, nothing need be said, for he states the problem and the remedy more eloquently than we can—Mr. Walsh:

"The task of preventing accidents in coal mines is a vast and unending one. During every working day of the year, tens of thousands of hazards are developed as a result of the orderly operation of the anthracite mines. The laws of Pennsylvania impose upon the 22,000 officials and certified miners the statutory duty to ferret out and remedy these injury-producing evils, and, strange as it may seem, a law violation exists until every one of these thousands of hazards is corrected. It should not be assumed, however, that all of the law violations are by any means of a negligent nature.

"While mining continues there will never be an end to the finding of hazards and law violations in coal mines, because the conditions requiring the application of law must be created before the law can be applied, and possibly as many as 30,000 of these hazardous conditions are developed each working day in the anthracite mines.

"The anthracite industry employs about 75,000 workers, and the law imposes upon the officials and the certified miners, 22,000 in number, certain specific statutory duties having for their purpose making the mines a reasonably safe place in which to work. How well this army of officials and certified miners has performed can best be seen in the fact that 73 pct of the fatalities that occurred in the anthracite mines during the year 1950 were caused by hazards that did not exist one hour before the accident occurred.

"There is no mystery attached to the occurrence of mine accidents, or the method by which they can be prevented. Better supervision, is a wonderfully efficient remedy."

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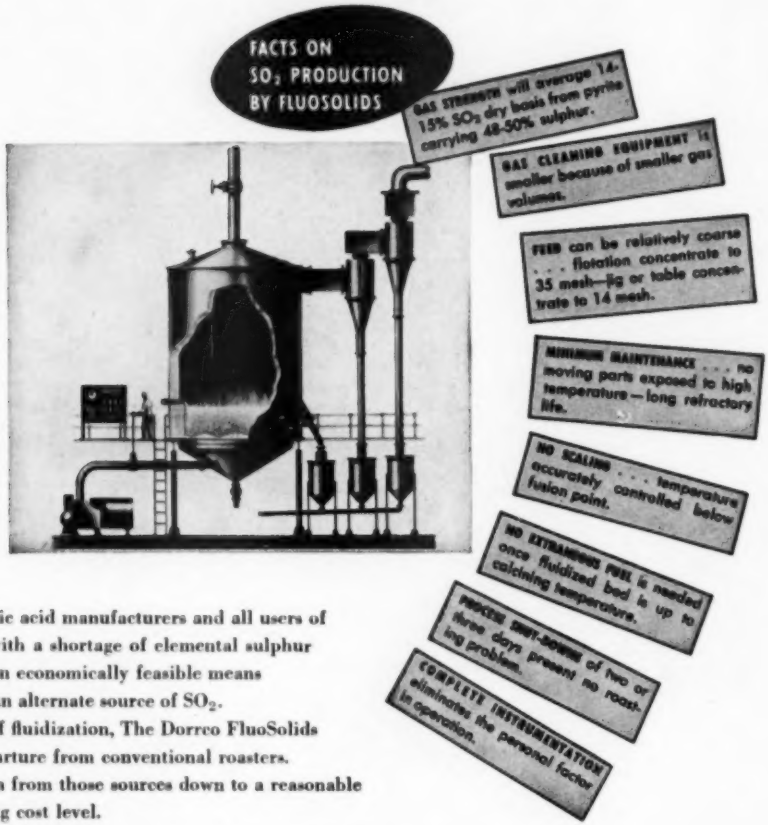
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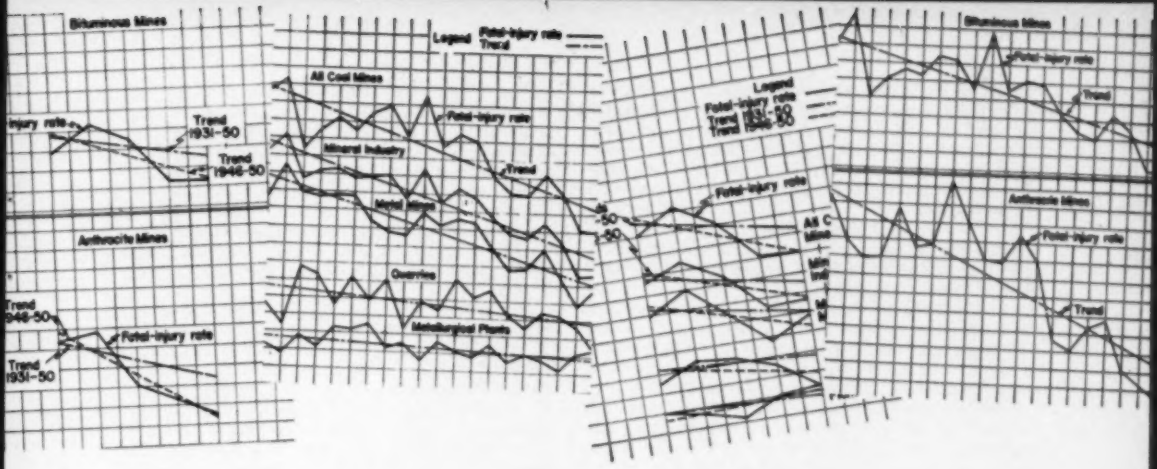


Mining Industry Continues With Lower Fatal-Injury Rates

by S. H. Ash

THE increasing need for the products of our mines, mills, and processing plants, the loss of mine manpower to plants other than those concerned with the mineral industry, and the drafting of our younger men into military service have created an acute situation in many places as to workers and production. This, in turn, has focused attention on the need for conserving manpower by conserving the health and life of our mining people.

Health

A world-wide tendency is noticeable in the mineral industry toward protecting the health of workmen, particularly younger workers, as well as older workers who can continue to produce in many places where manpower is lacking. This matter was emphasized at the Geneva Labor Conference for Coal Mines at the Committee's fourth session, held in Geneva, Switzerland, May 7 through 19, 1951.

Papers concerning the health of workers were presented at the annual meetings of the mining sections of the National Safety Council, the Coal Mine Inspectors' Institute of America, the American Public Health Association, and other organizations.

Safety Conferences

Without exception, the attendance at mining-safety conferences in widely separated places throughout the country during the year was the largest to date. What is needed at the safety meetings throughout the country is more-widespread attendance by those who can sell the idea to the workers that they are the focal point in accident prevention and that safety is not going to be achieved alone or even in part by laws, decrees, or commands, but by quiet influence, unconscious suggestion, and personal guidance. If these are acquired by the workmen, a safe environment and ample, capable supervision supplied by management, accidents will be fewer and less severe.

Many safety conferences attempt to develop the foregoing idea by enumerating and describing training programs for supervisors and workers alike. Nothing can replace a course in first aid for all persons employed at any mine or plant as an introductory and basic step to a successful safety program. Even in this, the shoe must fit the foot, rather than applying the order do this or else. In first-aid

MR. ASH is Chief, Safety Branch, Health and Safety Division, Bureau of Mines, Washington, D. C.

classes, safety instructors, management officials, and workers meet on common, sympathetic ground and the spirit of "talking things over" has its inception; safety meetings, discussions of accidents, safety committees, and prevention of accidents become interesting and worthwhile mutual problems that can be solved.

Roof Bolting

Since the beginning of mining, falls of material from the roof or back have caused over half of the mine accidents. Although frequency rates have decreased steadily in recent years, they are far from the irreducible minimum. A Roof-Control Section was organized within the Bureau of Mines in 1948 to conduct research and provide service to mine operators in solving their roof-control problems.

Roof bolting has proved to be effective; the Federal Bureau of Mines has sponsored many experimental installations in coal and noncoal mines and is keeping abreast of applied research in the laboratory and the field. Roof bolting has proved to be more applicable than was first suspected. Today, 570 mines are using this method of support, and it is estimated that roof bolting is increasing at a rate of 40 million sq ft of roof supported each month. The accident reports from several States, notably Alabama and West Virginia, continue to reflect favorable results.

Other activities of the Roof-Control Section are investigations of the relationship between mining methods and outbursts (bursts or bumps) and research on roof control of longwall-mining systems.

Electrical-Mechanical Research

Electrical-mechanical research by the Bureau of Mines is directed toward determining primarily whether or not the construction of electric machines and appliances will minimize or prevent gas, dust, and shock hazards. The increasing use of Diesel engines in mines appears to offer hope that transportation equipment powered by Diesel engines will remove much of the electrical hazard of explosions, fires, and shock. The Bureau has tested and approved a Diesel-electric shuttle car and a Diesel mine locomotive for mine use.

The Diesel-electric shuttle car was approved only for use in noncoal mines, that is, in mines that do not have to safeguard against ignition of gas. The principal safety feature of this shuttle car is its design.

The Diesel locomotive was approved for use in gassy coal mines. The electrical components are fitted with explosion-proof enclosures, and the engine intake and exhaust openings are fitted with flame arresters to prevent ignition of gas. The locomotive is equipped with an exhaust-gas conditioner and other features to limit the toxic constituents of the exhaust gases. To reduce fire hazards, the locomotive is designed and constructed so that the temperature of exposed surfaces will not exceed 400 degrees F.

For many years the Bureau's requirements for approval of motor-driven machines were not considered rigid enough to permit approval of mine locomotives and shuttle cars operated from cable reels. The requirements were amended in October to include additional safeguards. Three cable-reel shuttle cars are approved under the more rigid requirements, and a new type of permissible equipment has become available to industry.

Experimental mine-lighting systems were studied in connection with efforts to improve underground illumination. Because coal faces and ribs absorb approximately 95 pct of the light, the greatest opportunity for improving illumination appears to lie in increasing the reflectance of these dark surfaces by whitening them if practicable.

Research reveals that misfires in multiple blasting with electric blasting caps are not likely if enough current is available in the firing circuit. The prevention of misfires is largely a matter of using blasting units of adequate capacity.

Conveyors

The bottleneck in modern mining is transportation, whether from the face to the haulage level or from the haulage level to the surface. Conveyor haulage is increasing. Rubber-belt conveyors are being installed in main airways, and this introduces a hazard from fire that must be guarded against if disasters are to be averted. Serious hazards from the use of rubber-belt conveyors are the danger of fires from several causes, explosions from coal dust, and health hazards from air-borne dust.

Mine Fires

A list of accidents from fires at metal and other noncoal mines and quarries in the United States in which five or more lives were lost is given in *Information Circular 7493* by the Bureau of Mines. The

Safety conferences and first-aid training are methods of assuring safety consciousness among the workers and supervisors. Repetition is another sure method of implanting it.

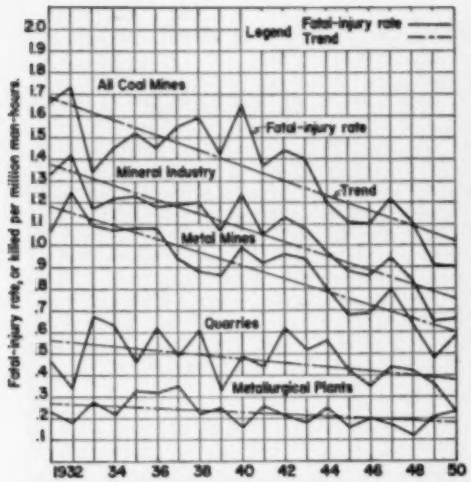


Fig. 1—Fatal-injury rates and trends for mineral industry and its branches, 1931 through 1950. The trend shows that the rates are down for all of the mining industry.

data give causes of the major accidents and describe the hazards and lessons that can be practiced to prevent mine fires. From 1869 (first available records) to date, 32 major fires have occurred in coal mines of the United States, killing 895 persons.

Mining operations should be conducted in such a manner as to remove the hazards or provide protective means so that mine fires cannot occur, and to do this, it is necessary to know what causes them. Most mine fires have resulted from the following causes:

1—Electrical failures.
2—Misuse of explosives.
3—Open lights and smoking.
4—Spontaneous combustion.
5—Welding or cutting steel.
6—Friction in rollers, power-transmission belts, and conveyor belts.
7—Misuse of fuel-burning equipment.
8—Combustible structures too near mine openings.
9—Incendiarism.

Because fires are a major disaster hazard in mines, the mining industry can well afford to take stock, review the causes of mine fires, and prevent them where the hazard is evident. Controlled ventilation, fire doors to sectionalize mines, and alert fire-prevention techniques are recommended, not only to prevent the destruction of lives and property if a fire inadvertently occurs but to be prepared to act promptly if a fire is started by sabotage or subversive activity, which by no means can be rejected as impossible. Personnel at every mine should recognize the dangers from mine fires and know what to do and what not to do in case of fire.

Fatal-Injury Rates

Figs. 1 and 2 show that fatal-injury rates continue downward and, with the exception of 1949, are lower than any time in mining history. The trend in all branches except anthracite mines and quarries, however, has been upward since 1949. From available statistics, the fatal-injury rate for 1951 in metal mines probably will be lower than in 1950. The anthracite and quarry industries continue to show what can be done by cooperation and applied safety effort to prevent accidents within large groups. The fatal-injury rate in anthracite mines for 1951 appears to be 0.83 per million man-hrs, as compared with 0.65 in 1950, the lowest rate attained by any one branch of the coal-mining industry.

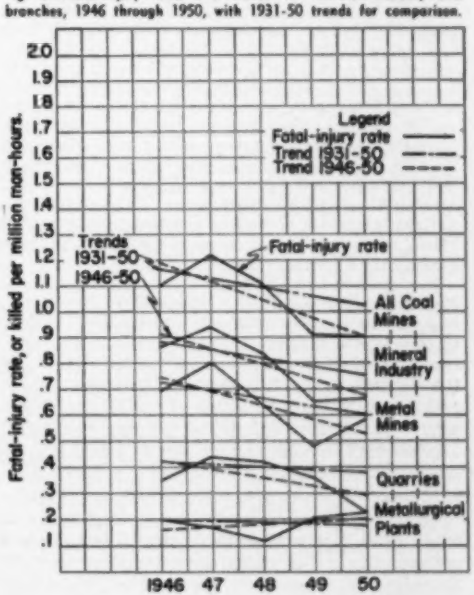
The year 1951 will be remembered in coal-mining history as a year having five major disasters, one of which (an explosion) on December 21 killed 119 persons. Three other explosions in bituminous-coal mines killed 33 persons, and one explosion in an anthracite mine killed 5 persons. The explosions in bituminous-coal mines were of electrical origin; a misused flame safety lamp caused the anthracite disaster. Four persons were killed by an inrush of water in a bituminous-coal mine.

Metal mines were free of major disasters during 1951.

As deplorable as disasters are, they continue to be from the same sources—explosions, fires, floods, and haulage-equipment failures. Moreover, just as the foreman is the key man in any safety program, training and experience are the tools with which he and his men must be equipped. Ample and practical supervision on the part of both officials and workers is the major remedy by which accidents can be prevented.

More persons are engaged in safety in American mines and other branches of our mineral industry than in any other nation today. All of them are building a safer industry; but, if they are to receive proper recognition and reward for the things they do, and if we in the mining industry are to improve our social economy and safety position, we must work in unity. No normal person or group of persons is antagonistic toward safety.

Fig. 2—Fatal-injury rates and trends for mineral industry and branches, 1946 through 1950, with 1931-50 trends for comparison.



At the Chino Division of Kennecott Copper Corp. at Santa Rita, N. Mex., safety committees have been established to maintain good practices in the mine and mill. This open pit operates its safety program through cooperation of all levels of employes and management.

SAFETY Stressed in Southwestern Copper Pits

SAFETY in open-pit mining receives a minimum of notoriety, but its significance in profitable and efficient operation is as great as in other methods of mining. Open-pit operators from Arizona, New Mexico, and Mexico, exchanging ideas on safety at a symposium during the recent annual meeting of the Arizona Section, AIME, discovered unanimity of opinion that the successful safety program depended on managerial and supervisory support with the vigorous cooperation of the employes.

Safety Organization

The operation of a safety program is dependent on the ability of management to project enthusiasm through the company organization. This requires a suitable method of communication to the supervisory and employee levels so as to approach them in a manner that will inspire cooperation. Safety committees comprised of various combinations of managerial, supervisory, and employe personnel are frequently used for this purpose.

The open-cut mine of the Chino Division of the Kennecott Copper Corp., Santa Rita, N. Mex., operates its program through management, a full-time safety engineer, and two safety committees, the Departmental and Shops Safety Committee and the Central Safety Committee. The Departmental and Shops Committee consists of one employe from each department and shop and the safety engineer. The selected members serve for a period of one month. This committee meets once a month, the only representative of management being the safety engineer because it is believed that employes will speak more freely when their supervisors are not present. The duties of the committee are to report hazardous conditions and make recommendations, but the committee has no authority to approve or disapprove. The Central Safety Committee is comprised of supervisory personnel from all departments and shops, and one member of each of the five bargaining organizations. Members of this committee are permanent and hold meetings soon after the departmental group to take action on recommendations of the latter. The committee system insures participation of all levels of personnel in the accident-prevention program, thereby stimulating safety consciousness.

The New Cornelia branch of the Phelps-Dodge Corp., uses three groups to conduct the safety program; the General Safety Committee, Division Safety Committee, and the operating-unit safety meetings. The unit safety meetings provide for active participation of employes in discussion of various phases of the program. Within these groups vigorous opinions regarding safety are often expressed and the workmen are critical if they recognize a digression from the adopted safe-practice procedures. These monthly discussions give the program vitality. The Division Safety Committee is composed of the division superintendent and the division department heads, and follows an established pattern in discussion of all phases of the safety program. The General Safety Committee, made up of the manager, division, and department heads, makes decisions on policies and procedures; it renders decisions on recommendations and suggestions. Management at the New Cornelia branch feels this system of safety committees and meetings has been active in advancing the program.

Safety Codes

In the open-pit mines of the Southwest, a safety code for operation or service of equipment, methods

Copper strip mine in Arizona, where safety plays an important part in economic operations. Proper safety apparatus and employe training will produce fewer lost-time accidents and fatalities.

of doing hazardous work, and use of protective clothing form a basis for accident prevention. These codes are not necessarily written, but, in cases, are handed along as rules-of-the-road.

At the Bagdad Copper Corp., open-pit at Bagdad, Ariz., truck drivers are given extensive training to qualify drivers for the adverse grades in the pit. At present, a driver-instructor breaks in new drivers, gives refresher training to older men, and serves as liaison between the pit and shop on the safety and operation of trucks. The new employe is introduced to the established operating code which is implanted in his mind by repetition.

New Cornelia has a flexible code of safe practices for the guidance of employes in the safe and efficient performance of their jobs. Code writing activity is a continuous process; major operational changes or the introduction of new equipment creates additional problems for which safe procedures must be determined.

Unlike some of the larger mines, Castle Dome and Copper Cities do not have a safety department. Experience has demonstrated that there is a fundamental principle underlying all effective accident prevention, namely that safety is an integral part of good operating practice and therefore a supervisory function. A reduction of accidents is not accomplished by indiscriminate discharge of careless men, necessitating the employment of new untrained men who may be equally careless. It is dependent upon the ability of the organization to obtain the proper spirit of cooperation from existing forces. The close relationship between foreman and workmen in the small company is exploited to indoctrinate workers in safety.

Safety Methods and Apparatus

The general safety program at Cananea Consolidated Copper Co., is designed primarily for accident prevention. It applies to all divisions of the operation, mining, concentrating, smelting, and miscellaneous departments. Each division carries on its own safety campaign, which is augmented by an over all program.

Each employe in pit production and repair shops is provided with safety equipment consisting of hard hat, hard toed shoes, goggles or safety glasses, and gloves. The foremen are made responsible in enforcing the use of protective and safety equipment and each foreman is instructed to insist on doing the job safely or not doing it at all. Further to encourage safety, a system of group safety awards, begun in 1949, has contributed greatly toward a reduction in lost time accidents. The men were divided into groups composed of those working closely together on the same type of work; for example, those men engaged in shovel loading and haulage make up one group; those engaged in drilling and blasting make up the second group; and those on truck maintenance make up another group. For completion of 4000 shifts without a lost time accident, each man in a group receives an award such as a pocket knife, cigarette lighter or some substantial award. In this manner each man is encouraged not only to look out for his own safety but to be vigilant of his fellow workers so that no man in the group may suffer an accident. An accident to any single man in the group which causes lost time in excess of four days means that the group must start over toward accumulating the 4000 shifts to earn the group award.

Safety practices in the open pit at Inspiration Consolidated Copper Co., Inspiration, Ariz., are designed to prevent accidents at or around the various operations. All employes working in the open pit are required to wear hard hats, hard-toed shoes, and employes on the night shifts are required to use cap lamps. Head and foot injuries are uncommon in the operation. Cap lamps have contributed a great deal to the safety of night operation. Fire extinguishers are maintained in all trucks, and shovels are conveniently located in the truck repair shop. First-aid kits are provided at the scenes of greatest activity.

First-Aid Training

At many of the open pits it has been found that first-aid training serves a dual purpose. It not only trains the workers to reduce severity of accidents, but makes them more safety conscious.

Training of employes in first-aid procedures has managerial support at the New Cornelia Branch The prompt application of knowledge of first aid fundamentals may control the severity of an injury, but such training is also considered of value to focus attention on safety.

Nothing can replace first-aid courses for all persons employed at any open pit or mine as an introductory step to a successful safety program. In first-aid classes, safety instructors, management, and workers meet on a common ground and the spirit of talking things over is created.

Ultra-Fine Grinding and Classification With Fluid Jet Pulverizers

by M. F. Dufour and J. B. Chatelain

FLUID jet mills employing a compressible fluid to grind materials to the sub-sieve range, was developed in the early 1930's, and the subsequent acceptance of these units was the result of their ability to pulverize solids that, because of physical properties, could not be processed by other attrition methods, such as ball, pebble, and hammer mills. Also, the fluid energy pulverizer is the only grinding device that will discharge continuously a 5-micron or smaller product. Simultaneous ultra-fine grinding and classification is the widest application of this unit. Jet mills may be used to advantage to reduce particle size of solids suspended in liquids, for coating one material with another, and effecting chemical reactions and changes in physical characteristics other than a reduction in size.

Materials that can be processed by this grinding method include nonmetallic and metallic minerals, ores, metals, pigments, insecticides, fungicides, pharmaceuticals, plastics, dyes, and numerous other organic and inorganic products.

Grinding principles utilized in the fluid mills available on the market are basically the same, the major difference being in the shape of the grinding and classification chamber. Grinding in fluid jet pulverizers is accomplished by collisions between particles of the material being ground with the energy being supplied by compressed fluids entering the grinding chamber at high velocities.

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The "Micronizer" is a fluid energy mill employing a relatively flat cylindrical grinding chamber. Fig. 1 shows general construction details. The mill shown is equipped with a venturi type feeder using a compressed fluid to inject the feed into the grinding chamber.

The energy consumed in forcing the feed into the grinding chamber against the mill pressure is, in some cases, as much as 40 pct of the total required for processing. Recent improvements, using a compression screw feeder as shown in Fig. 2, can reduce this to 1 or 2 pct of the total energy requirements.

Grinding

Grinding, as carried out in the Micronizer involves the action of gaseous jets emerging from nozzles located in the peripheral wall of the grinding chamber. The axes of the jets are tangential to a circle of smaller diameter than the grinding chamber as is shown in Fig. 1, and the gases entering the chamber maintain a high rotational velocity. The feed introduced into the mill is accelerated by the rotating gases and is thrown to the outside where a portion, coming in the zone of action of a nozzle, is further accelerated tangentially and inward. Violent collisions occur between the particles so accelerated and other particles circulating at a slower rate in a thin band near the periphery. It is estimated over 90 pct of the reduction in Micronizers occur as a result of impact within a fraction of an inch of the orifice. Observations also indicate that size reduction is independent of any interaction with the walls of the

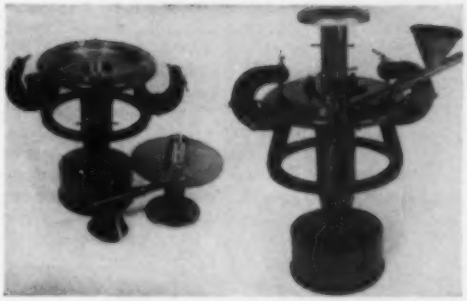
Trends for paint grades of talc are in the direction of finer grinding, either by two-stage classification or fluid energy grinding. One of the largest developments in fluid energy grinding of talc ores is being performed at the Gouverneur Talc Co.'s mine and mill near Balmat, N. Y., subsidiary of the R. T. Vanderbilt Co., New York.

The ore is mined by conventional and modern methods of stoping and haulage, passed over grizzlies and hoisted through a 500-ft vertical concrete-lined shaft. The ore is crushed on the surface from 10-in. down to suitable mill feed. The fluid energy circuit consists of 14 C. H. Wheeler fluid energy mills, which employ heated compressed air as the fluid, or energy agent. Superheated steam may also be used as a fluid but due to operating difficulties and climatic conditions, it was abandoned in favor of compressed air. The air entering the mills is at high pressure and temperature. The air is compressed by an Ingersoll-Rand air compressor having a 1750 hp synchronous motor. The air leaves the compressor without an aftercooler, then to an Alcorn air heater, which heats the air before entering the mills, air heater is fully automatic. The mill feed is injected into the mill by the use of compressed air against an internal mill pressure.

Ore is reduced in size within the mill by both impact of particles with each other and against the walls of the mill. The material when reduced to the size to meet the conditions within the mill exits with the spent fluid and enters two cyclones connected in tandem, each of which produces a product of different particle size. The air leaving the second cyclone enters a cloth type collector for separation from the air a third and final product of extremely fine particle size.

chamber. The fines resulting from the series of impacts are carried by the rotating gases in an inwardly spiraling path towards the central outlet. Since the solids travel in the vortex at substantially constant tangential velocity, which results in increasing centrifugal force, they are subjected to an intense classifying action as they approach the outlet, the centrifugal force returning oversize particles to the grinding zone while the finer particles are carried by spent grinding fluid to the outlet. The successful performance of this classification barrier is one of the main reasons for the wide acceptance of the fluid jet pulverizer.

The fluid carrying the fine solids from the mill is passed through a cyclone type collector which may be an integral part as shown in Fig. 1 or separated from the mill as shown in Fig. 3. If precautions are taken to preserve the rotational speed of the gases leaving the mill, no additional energy is required to obtain collection in the cyclone. The efficiency of collection depends on the specific gravity and particle size of the solids, as is the case with all cyclone type collectors. Practically all commercial installations operate with a primary collector efficiency of above 85 pct, even where the product has an average particle size of 2 microns or less. Normally the exhaust from the cyclone collectors is cleaned by a bag filter or wet collector, but in a few applications all the product is collected in bag collectors.



Assembled and unassembled views of a 15-in. Micronizer, showing inlet for the energy source, venturi feeder and grinding chamber.

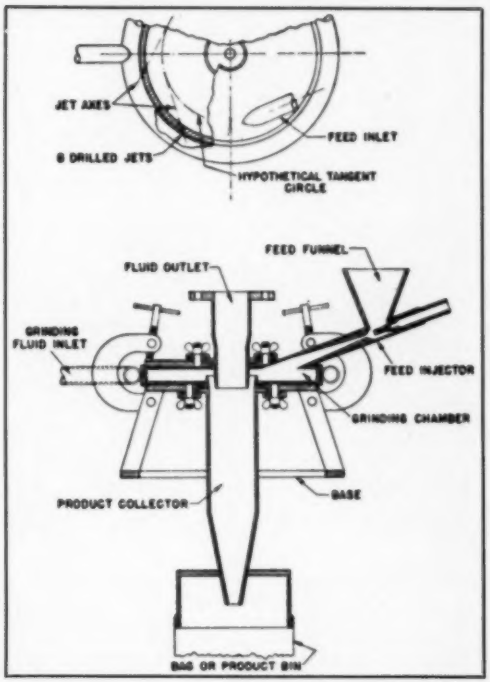


Fig. 1 — Drawing, above, showing major features of construction of Micronizer unit equipped with venturi-type feeder. Collisions within the thin circular chamber are responsible for the size reduction.

Grinding energy for most commercial installations is supplied in the form of compressed air at 100 lb per sq in. and at temperatures ranging from below atmospheric to 700°F, or steam at pressures from 100 to 200 lb per sq in. and temperatures ranging from 400°F to 1000°F. The volume of compressed fluid required to grind a unit weight of material is, in general, proportional to the kinetic energy of the fluid supplied.

The major application for the fluid energy mill has been in the field of dry grinding and classification of solids to particle sizes below the sieve range, but in this field of general use there are limitations.

Generally speaking, solids that are not frangible at the temperature existing in the grinding chamber cannot be reduced satisfactorily. These include most fibrous materials, a number of metals, and some waxes and similar organic materials. Hard abrasive materials are being ground to average particle sizes of less than 5 microns, but in cases it has been impossible to limit the contamination from abrasion of mill parts to the permissible range.

Capacities and Sizes

Micronizers are rated according to the inside diameter of the grinding chamber which varies from 2 to 30 in. The height of the grinding chambers at

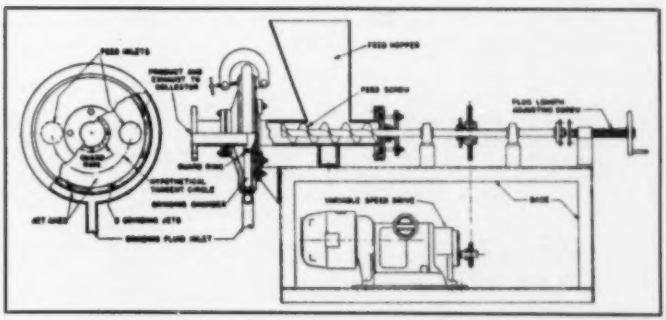


Fig. 2 — Fluid jet pulverizer which is fed with a pressure screw feeder. This unit does not require a compressed fluid to introduce the material against the mill pressure.

the periphery may vary from ⅜ in. to 2 in. The number of grinding jets will usually range from 3 to 24, this and other features of construction varying according to the material to be ground and the results desired.

Table I shows the capacity of different sizes of Micronizers and energy requirements for grinding a number of materials. The majority of materials was purposely chosen from the class of nonmetallic minerals and the data are all for simultaneous grinding and classification to the lower micron sizes. The capacity of Micronizers will vary from ½ to 4 lb per hr for the 2-in. mills to 500 to 3000 lb per hr for the 30-in. units.

The information given in Table I for talc, iron oxide, cryolite, barytes and Fuller's earth all show steam as the source of energy. Steam is normally used since it is a less expensive source of energy and the materials listed are not harmed by the temperatures involved. However, there are installations processing these materials using compressed air at various temperatures. Another advantage realized with steam is that a higher capacity can usually be realized from the same size mill with steam instead of air.

Table I. Grinding Data for Materials Processed in the "Micronizer"

Material	Mill Diam, In.	Grinding Medium Type	Grinding Medium Flow	Solid Feed Rate	Approx. Avg. Particle Size
Titanium dioxide	30	Steam	4000 lb per hr 5,687,200 ft-lb	2250 lb per hr	Less than 1μ
Sulphur	24	Air	1000 cfm	1300 lb per hr	3-4μ
Talc (varies)	30	Steam	4000 lb per hr	2000 lb per hr	2μ
Iron oxide pigment	30	Steam	4000 lb per hr	1000 lb per hr	2-3μ
Cryolite	30	Steam	4000 lb per hr	1000 lb per hr	3μ
Barytes	30	Steam	4000 lb per hr	1800 lb per hr	3-4μ
Fuller's Earth	20	Steam	1200 lb per hr	600 lb per hr	3-4μ, 15μ top
Anthracite Coal	20	Air	1000 cfm	1000 lb per hr	5-6μ
DDT (30 pct)	24	Air	1000 cfm	900 lb per hr	5-6μ
Procaine-Penicillin	8	Air	100 cfm	10 lb per hr	3μ, 20μ top

Air—100 lb pressure—60°F.
Steam—150 lb pressure—550°F.

Operating Costs

The major processing cost in fluid jet pulverizers is energy. The cost of energy will vary depending on the material processed, the size of the feed, product specifications and the cost of fuel or electrical energy. Cost will range from $1 to $10 per ton for pulverizing most materials to the sub-sieve range. The above costs appear high when compared to the cost of particle size reduction in mechanical impact equipment, but an example will show that this is not the case when the new surface created is considered. If a 30-mesh (590 microns) material is reduced to 325 mesh (44 microns), which is considered fine grinding for most mechanical pulverizing equipment, the number of particles is increased by 2400 and the surface area is 13.4 times greater. If the 30-mesh material is reduced to an average particle size of 5 microns, the number of particles is increased by 1,643,000 and the surface area by 118. Therefore, based on the generally accepted theory that the energy required for grinding is proportional to new surface created, it can be seen that 8.8 times more energy is required to grind a particle from 30 mesh to 5 microns as compared to grinding to 44 microns.

Size of Feed

Solids 4 mesh or larger are being processed in the larger sizes of jet mills. The only limitation is that the feed be of such a size that it will not plug the feed mechanism. However, it is normally advantageous to precrush to at least 10 mesh and in some cases to 100 and 200 mesh. One advantage in reducing the size to the minimum practical with mechanical equipment is that this normally results in the most efficient overall operation. Grinding solids to the sub-sieve range is increasing in importance as industry finds that advantages resulting from increased surface area and smaller particle size allow a reduction in overall cost of producing a finished product with improved properties. Jet mills are ideally suited to production of these materials and is the field where they are most widely used.

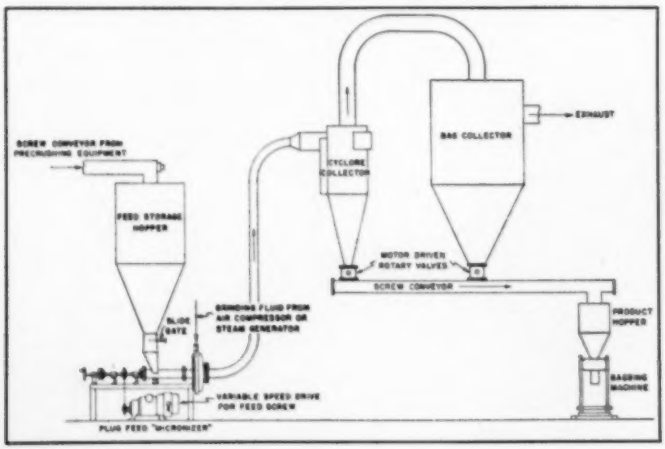


Fig. 3—Flowsheet of a fluid energy grinding unit with separate cyclone collector. The mill is equipped with screw feeder to feed the material and regulate the rate. Five-micron products, or less, are easily made with this machine.

World's Largest Asbestos Producer Uses Block Caving and Concreted Slusher Drifts

Circular concrete headframe, No. 1 shaft of the Jeffrey mine of Canadian Johns-Manville Co., Ltd., Asbestos, Que. Headframe was constructed in a continuous pour with sliding forms.

by Karl V. Lindell

THE Jeffrey mine of the Canadian Johns-Manville Co., Asbestos, Que. has operated for a number of years, supplying the parent company, Johns-Manville, raw material for asbestos products. The mine is situated in the 70-mile-long serpentine belt running from Asbestos, Que. on the west to East Broughton on the east. This district is the most important producing area in the world of quality spinning-fiber asbestos, with 11 operating mines supplying 70 pct of the world's needs. The Jeffrey mine, located 90 miles from Montreal and approximately 35 miles southwest of Thetford, accounts for 50 pct of the Canadian asbestos output.

Production from the Canadian serpentine belt is the chrysotile asbestos, which is considered the best available for spinning. Consumers in the United States prefer the clean, fine fibers of the Canadian chrysotile of the type produced by the Jeffrey mine.

Mining

In 1881, when the mine was first opened, all work, including drilling, was done by hand. The deposit, located on a hillside, was first worked as a sidehill cut by hand drilling 15-ft holes and blasting with black powder. The fiber was chiselled from the broken rock. Hand derricks, later powered by horses, were used to raise the rock as the pit deepened. Eventually, the pit became so deep that derricks were, by necessity, replaced by aerial tramways.

In the late 1930's, further deepening of the open pit, with the resulting constriction of the pit bottom brought about the need for altering the mining system then in use. The high ratio of overburden to be handled to mill ore mined and the value of surface structures, not company property, that would be destroyed in expanding the area of the pit, definitely indicated vertical rather than horizontal expansion would have to be used. One exception to this was that a small area on company property to north and east of the pit could be opened up.

The mining problem then resolved itself into two phases; make the small lateral expansion and as rapidly as possible choose a mining method that would expand the mine vertically.

Several methods for vertical expansion were studied and rejected. Methods considered included shrinkage stoping, block caving, glory holing, and various modifications of open workings.

Block caving was finally accepted as the most suitable method of underground development. Cost factors and production were important factors in the decision, also, the asbestos mines in the Thetford area were already having success with this method.

Original plans provided for the lateral expansion of the open mine where possible and beginning the underground operation. This program was based on the open pit being worked out in 1954, with the underground development and production gradually increasing until the open mine was depleted. In 1949, the plan was changed to allow underground development to progress at maximum so that underground output would be 550 tons per hr by 1951. Any deficit in ore would be mined from the open pit. This change permitted extending the life of the open pit through 1964 and in this time would make available a large quantity of readily attained ore.

Block Caving at The Jeffrey Mine

Plans for the underground operation were formulated on a 12,000-ton-per-day basis, and the adaptability of block caving methods to high production presented a desirable situation. As shown in the main-haulage plan, a continuous loop was laid out for smooth operation of the underground railroad from the loading points to the underground crusher. A feature encountered at the Jeffrey mine, as well as other asbestos mines using caving methods, is the large size of caved material, requiring that ore be crushed before hoisting.

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The system adopted for block caving was based on mining 200x200-ft blocks, cut off on all sides by a series of sublevels, with ore transfer to the haulage by slusher drifts. The main haulage level is made up of a continuous loop system of drifts and crosscuts, 8 ft wide by 9 ft high inside the steel or concrete. The No. 1 and No. 2 main haulage drifts are parallel on 100 ft centers through a central pillar in the orebody with loading crosscuts normal to them every 200 feet. These in turn are all joined at their most remote points from the shaft by a drift on the fringe of the orebody. The grade in all headings is ½ pct grade leaving the No. 1 shaft.

Between No. 2 shaft and the orebody, safety bays are provided in haulage drifts where other refuge is not available. The necessary stations to service the mine equipment and men such as a storeroom, fitters' shop, d.c. substation and lunchroom are located in the central pillar. Within the orebody, but outside the central pillar, are unit substations for the slusher hoists, powder and cap magazines, an oil storage room, fan station and ventilation drifts and crosscuts. On the edge of the orebody, at the junction of No. 1 main haulage and 1-3 crosscut, is the dispatchers' station, and near No. 1 shaft, the pumproom and sumps, tipple station, aggregate station and charging station complete with overhead crane and pit to service the electric locomotives, are situated.

Main haulageways are driven with a twin-boom hydraulic drill jumbo mounting drifters using 1-in. quarter octagon drill steel and detachable bits. The broken muck is loaded into tipple cars with an electric Conway loader, but air-driven model 21 Eimco rocker shovels and 2-ton cars are also used for small work. Stations are normally driven using the above equipment, or with an air-driven self-propelled Drillmobile mounting a hydraulic boom where track is not available.

Headings on this level are left untimbered but combinations of gunite, timber and concrete are used where support is needed. The two main haulage ways are concreted from the sumps to the d.c. substation because of ground conditions outside the orebody, and gunited or timbered within the orebody.

Haulage on the 750 level is on 36-in. gage track of 85-lb rails laid on steel ties, with six 15-ton combination trolley and battery locomotives pulling trains of ten 200-cu ft solid-body tipple cars with rotary couplings. Two 3-ton battery locomotives for the 40-cu ft rocker-dump cars, 5 passenger cars, and miscellaneous cars complete the rolling stock list.

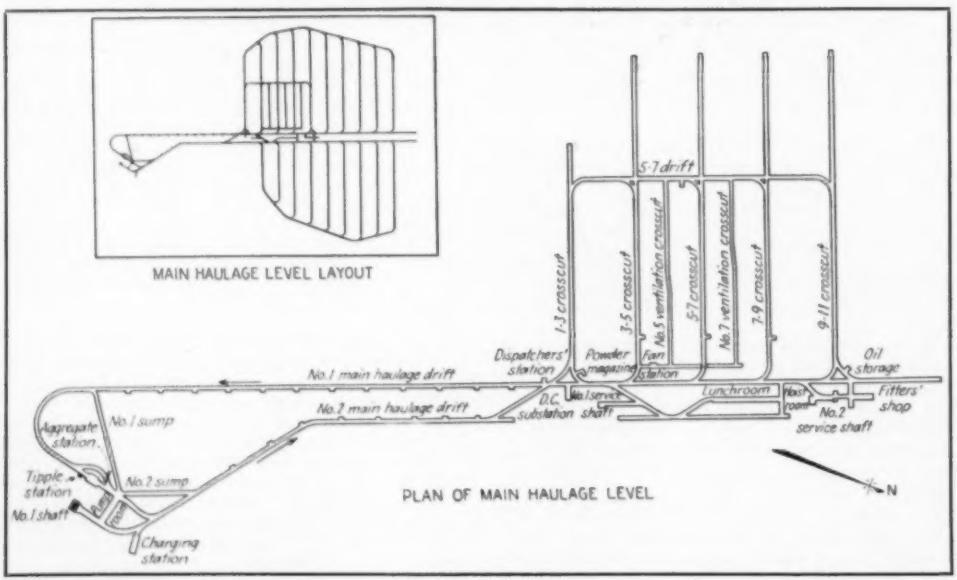
All train movements are controlled by the dispatcher located at the intersection of No. 1 drift with 1-3 crosscut and No. 2 crossover. Communication between the dispatcher and the 15-ton locomotive is by means of two-way radio telephones. There are telephones installed at strategic locations in case of radio failure and for the use of the 3-ton locomotives.

Trolley is not installed in any crosscut where there is loading from chutes or slusher drifts, in these areas locomotives lower their trolley poles and use batteries.

The first step in the development of a 200x200-ft block is to provide a means of access to the sublevels that form the cutoff around the stope. To accomplish this, service shafts are placed at strategic locations throughout the orebody, each shaft servicing a group of blocks. At present there are two shafts in operation, both within the central pillar of the orebody.

The No. 1 service shaft was sunk from the 615 level in the open pit and bottomed on the 750 level, a total depth of 622 feet. It is wooden timbered, concreted in weak ground, and has three compartments, a cageway for a 10-man cage, a pipeway and a ladderway. No. 2 service shaft is an internal shaft between the 750 level and the 9th sublevel. It is steel timbered, concreted where necessary, measur-

Layout of the main haulage system of the 750 level, showing location of major installations of the underground mine plant. All haulage drifts on this level are 8x9 ft and supported with combinations of steel, concrete, and gunite.



ing 5 ft 6 in. x 10 ft 7 in. and has a cage compartment for a 10-man cage, rope, pipe, and ladder compartments. It was excavated by drifting to it on each level from No. 1 service shaft, raising a 4x4-ft raise by diamond drilling from level to level, then slashing it to size and timbering from the top down. The hoist for this shaft is located in a steel timbered and gunited room on the 750 level.

From the service shaft, 4x7-ft sublevel drifts are driven at 40-ft vertical intervals to the block under development. The 200x200-ft block is then perimetered on these levels with the drifts. Drilling is done by jackleg machines using chisel bit ⅞-in. hexagonal steel, and the broken rock is slushed by 36-in. scrapers and 10 hp slusher hoists to 4x4-ft raises driven by diamond drills on each corner of the block. About 33 pct of these headings must be supported by steel timbers, but 95 pct of this is recovered before the block is caved. The corner raises terminate in chutes located in the 750 level crosscuts, from which cars may be loaded. Fringe drifts will only be used for initial caving after which modified panel caving will be applied.

In the first stages of the caving, it has been necessary to cut off the blocks all around, to start the cave working. After sufficient area has been disturbed, a panel caving method, eliminating a large portion of the cut-off mining will be used. A full-panel caving is a system whereby adjoining panels are undercut and caved in a regularly retreating manner, the ore drawn from the stope in the same manner. In this method, perimeter cut offs are not used, except in special cases.

Each block is laid out with eight slusher drifts, four on the north and four on the south sides, spaced on 50-ft centers along the crosscuts. Slusher drifts are spaced in blocks on opposite sides of the crosscuts so that although they are on 50-ft centers for any one block they are on 25-ft centers along the crosscuts—the Nos. 5, 6, 7 and 8 of one block on one side, and the Nos. 1, 2, 3 and 4 slusher drifts of adjoining block on the opposite side of the crosscut.

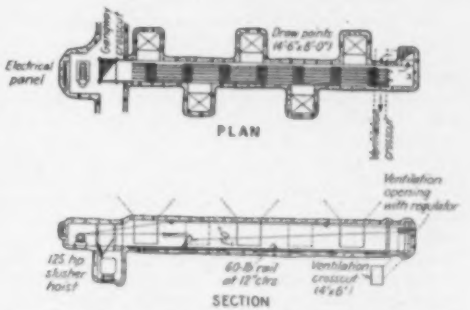
The haulage drift walls are concreted to the bottom of the caps for the length of one or two blocks by placing 3/16-in. steel from plate set to set using 5-in. I-beams for walers and pouring concrete to the rock behind these forms. The walers and plates are removed, a flume is excavated, concreted and covered with checkered plate.

Slusher drifts are driven normal to the haulage crosscuts from the lip plates at 1 pct grade half way across the block. Working off the muck for the top half of the round, jacklegs are used for excavation. The excavation is a minimum of 10 ft wide x 13 ft high, allowing 18 in. of concrete in the walls, 1 ft on the floor, and 3 ft in the back, around a finished drift 7 ft wide x 9 ft high. The 60-lb rails laid in the floors have their ends bent 45 degrees so as not to be torn up by the scrapers. There are two lengths of slusher drifts; the Nos. 1, 2, 3, and 4 on the south side of a block are 103 ft long; while the Nos. 5, 6, 7 and 8 on the north side are 120 ft long.

After the floor is poured the first round is slashed out of each drawpoint and the ventilation stub excavated at the end of the drift. Drawpoints are on 35-ft centers on the same side of the drift or staggered on 17 ft 6 in. centers from side to side, the No. 1 drawpoint being 12 ft 6 in. from the center line of the crosscut.

Panel and truss type aluminum forms with quick acting clamps for assembly are used to concrete the slusher drifts. The 12-in. batter in the top corners of the drifts is built in the forms, and recessed boxes and hanger bolts for the return cables are placed as the forming proceeds.

After completing the concreting of the slusher drift, the drawpoints are excavated to 12 ft vertically above the slusher drift floor and for a distance of 16 ft along the 45 degree footwall. This part of the



Drawing of typical concrete slusher drift showing drawpoints, ventilation connection, and the concreted haulageway.

drawpoint is poured after restanding 7 trusses of the drift forms and clamping aluminum drawpoint forms in place. The finished drawpoint opening is 8 ft long by 4 ft 6 in. high.

Upon finishing the concrete work, the slusher drift is ready for the final stage of development. A slot 5 ft wide x 25 ft long on the center line of the drawpoints is driven normal to the slusher drift, to the undercut level floor.

Undercutting

Simultaneously with the development on the sublevels and 750 level, undercutting by diamond drilling is started on the 1st sub or undercut level. Three 7x8-ft undercut drifts are driven 200 ft through the block directly above and parallel to Nos. 1, 2 and 3 slusher drifts. Diamond drills are set up in the undercut drifts and fringe drifts to cut out a 200x200-ft slice 7 ft high and to shape the drawpoint bells. This entails drilling approximately 50,000 ft of 1⅝-in. hole, some drilled with blasthole bits and some cored, depending on the ground. The spotting and lining up of these holes must be accurate and requires constant checking by the supervisory and engineering staffs. A careful record is kept of the loading and the loaded length of each hole, along with the number of sticks of powder compressed in that length.

Priming is carried out according to a carefully prearranged layout using short period delay electric caps. It is most important to follow the priming plan as the shot is planned to break the ground within the cone of each drawpoint and immediately above it as high as the undercut roof, sideways and downwards into the excavated slot.

Connections are not made until all loading and priming has been completed and the records checked to balance the series. Prior to connecting up No. 10 plastic covered wire is strung on insulators through the first sublevel drifts to the top of the ventilation raise into a safety switch.

Production

Production is attained by loading direct from the slusher drifts into 200-cu ft tipple cars with slusher hoists and 72-in. scrapers. Slusher hoists have rope speed of 260 ft per min, are equipped with electrically operated remote controls with air-actuated instantaneous clutches, and are powered by 125 hp 2200 volt a.c. motors. Both 72-in. rigid and folding type scrapers are in use. The 1¼-in. return cable passes around an 18-in. tail pulley mounted on a 14-in. reinforced H-beam tail post.

Each slusher drift is illuminated by a sealed beam floodlight mounted in front of the slusher hoists. The loading point is the full width of the slusher drift and the steel loading lip projects 6 in. inside the near edge of the tipple car positioned in the haulage drift. On the opposite side of the crosscut to the loading lip a steel spill door is suspended to deflect any material overshooting the car. Cars are loaded to 8 or 9 tons by two or three passes of the scraper.

Secondary blasting of hang-ups and large chunks in drawpoints in slusher drifts is done as required. Drawpoint hang-ups are broken by a bomb of 5 to 50 sticks of 1¼x8-in. 75 pct Forcite attached to the end of a rolled sheet metal blasting pole placed in the desired position in the drawpoint. Rocks which have passed through the drawpoint but are too large to load are drilled and blasted in the slusher drift. Secondary powder consumption for the ten months of production in 1950 averaged 0.48 lb per ton of rock produced from the slusher drifts. However, as a larger area is opened up this figure shows a tendency to decrease.

Prior to February 22, 1950 all production was from development work and ran about 600 tons per day with a maximum of 1434 tons attained one day. In February, a quarter of the first block was caved, and total production increased from an average daily figure of 980 tons in March to 5000 tons daily in December.

Mine Plant

The development of this elaborate underground mining program required the construction of an equivalent mine plant, including shaft and its auxiliaries, underground crusher, and mine facilities to operate the mining equipment.

The main production shaft is steel timbered, concrete lined and is divided into five major compartments. The shaft compartments provide for two 10-ton skips operating in tandem, a 50-man cage, cable, pipe, and counterweight. The steel sets are 6-in. 22.5 lb H-beams, on 6-ft centers.

The shaft headframe is unique in the mining field, inasmuch as the concrete headframe structure was constructed with a sliding form, in a continuous operation. The main structure was built in one pour which took almost 15 days to place the 796 cu yds of concrete needed. The headframe is 115 ft 6 in. high with a 12-ft penthouse, and 30 ft in diam.

The headframe houses two 10-ft rope sheaves and two 10-ft cage and counterweight sheaves. The two sets of sheaves are mounted on separate concrete slab decks 100 ft and 74 ft, respectively, above the shaft collar. One of the empty segments of the 30-ft diam headframe, outside the shaft proper, houses a stairway and elevator well. The segment on the collar house side serves as a hoistway for repair work on the sheaves.

Hoisting equipment at the No. 1 shaft consists of two Ward-Leonard amplidyne controlled hoists, one a 200 hp cage hoist, the other a 1000 hp skip hoist. The hoists have standard Ward-Leonard equipment, with the addition of amplidyne and electronic controls. These two controls, working together, control maximum voltage or speed, maximum current and torque, and rate of change of acceleration. It is possible, with this system, to throw the controller from full hoist to full reverse without tripping circuit breakers or causing any discomfort to those riding in the cage or skips. The cage and the skips are equipped with pneumatic guide rollers. Elevator control of the cage is made possible by the amplidyne and electronic control, for manual control of acceleration is unnecessary. Station control of the cage is a refinement which permits the cage to be loaded or unloaded at either the 700 or 750 level and sent automatically to surface unattended.

The ore hoist is operated by remote control with amplidyne control. At the loading pocket there is a control station with push buttons for "run", "stop", and "emergency stop".

A circular reinforced concrete surge bin of 300 tons live capacity was erected on the west side of the headframe. The 20-ft dia. bin is provided with 12-in. circular shelves spaced so as to build up a lining of rock fines to protect the walls against impact and abrasion.

A steel frame collarhouse, with standard and 36-in. gage tracks entering the building and 10-ton electric crane handling heavy material, was constructed after the shaft was completed.

The changehouse has a structural steel frame with brick and concrete block walls and partitions. The change room is of the hang-up and locker type, with 500-man capacity. Ventilation is provided by an 8600-cu ft per min fan. Both showers and Bradley type wash fountains are provided. The building also has a 30-locker foremen's dry room, an 8-locker officials' dry, a first aid room and offices.

Underground Plant

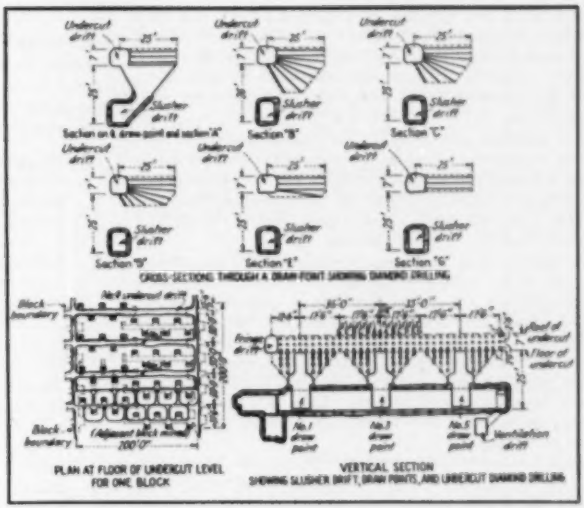
A rotary tipple for dumping two 200-cu ft cars into either the waste or ore bins was installed on the 750 haulage level at the No. 1 shaft. The tipple, which is actually two 1-car tipples operating in unison, is located in a concreted station, with the 130-ton waste bin on one side and the 260-ton ore bin on the other. Both storage units are lined with steel plates anchored in the concrete walls.

The crusher room, located on the 816 level, is separated from the shaft by a 35-ft pillar. It is constructed of reinforced concrete construction, three walls and the arched back being a minimum of two ft thick and the fourth wall, which has the three crusher feed openings, a minimum of three ft.

Two 48x60-in. jaw crushers are installed in the crusher room with 200 hp drives, fed by vibrating grizzly feeders from the tipple ore and waste bins.

The No. 1 crusher feed comes from the tipple waste bins, and the by-passed fines and crushed material discharge into a bin below the crusher. The No. 2 crusher feed comes from the tipple ore bin, and the by-passed fines and crushed material may be discharged either into the waste bin under No. 1 crusher or the ore bin. The No. 3 crusher, which will be installed when required, will draw its feed from the same ore bin as No. 2 crusher, and will discharge into the ore bin beneath its planned location. A 20-ton electric travelling crane is set up to service all three crushers.

Ore and waste bins are identical, set under the No. 1 and No. 3 crusher pits. They have a live capacity



Undercutting method used at the Jeffrey mine for a 200x200-ft block. Drawing shows pattern of diamond and percussion drill holes to shape drawpoints as well as remove the 7-ft slot. Drawpoint spacing is 35 ft, with finger raises staggered on opposite sides of the slusher drift.

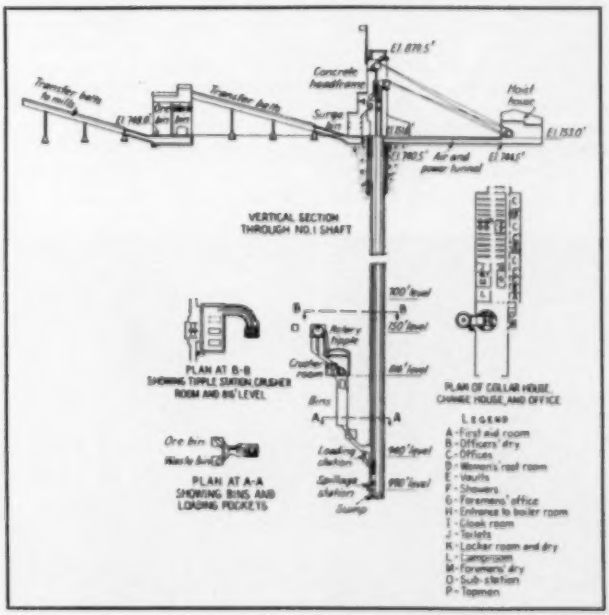
of 350 tons each, are steel timbered, lined with vertical 65-lb rail on 1-ft centers, and concreted flush with the wearing surface of the rail. From each bin, a short steel lined slide conducts the crushed material to the top of the loading pass. This loading pass delivers either waste or ore to the measuring pockets. Two 10-ton capacity measuring pockets are installed in a concreted shaft station on the 940 level.

Auxiliary Underground Plant

Power is taken underground at 2200 volts a.c. with four 3-conductor, 250 MCM armored cables in parallel. These leave the shaft at the 750 station and run to a main bus in the 750 pump room, containing a transformer vault and control panels.

The d.c. substation is off No. 1 main haulage just in the orebody. It is a concreted room in which three 215 K.W. motor-generator sets are installed to provide 250 volt d.c. power for the locomotive trolley and tipple drive. Lighting throughout the mine is 110 volt a.c.

The unit substations for the slusher hoists are 300 k.v.a. units, complete with Askerel filled transformer and secondary distribution switchboard consisting of one main circuit breaker, four branch breakers, and four slusher feed plug-in receptacles.



General drawing of the No. 1 shaft and the various auxiliaries associated with it. Relative position of the underground crusher is shown, with the facilities for loading the 10-ton skips.

These units are capable of supplying four 125 hp slushers at one time with 550 volt a.c. power. Although the capacity of each unit is limited to supplying power to four slusher hoists at a time, the unit is located so that any four of eight may be plugged in.

Compressed Air and Water. Compressed air is taken underground from the compressor in the No. 1 shaft hoistroom through an 8-in. air line down No. 1 shaft, and from the open pit through a 4-in. line in No. 1 service shaft. It is distributed in the mine with 6-in., 4-in., and 2-in. pipes.

Pumping. The 750 level is drained by concreted ditches 18 in. wide x 14 in. deep, running along the wide side of all the drifts and crosscuts. These ditches drain into the No. 1 and No. 2 sumps. Baffles settle out asbestos fibres and fines. An 8-ft dam fitted with control valves at the lower end of each sump allows either or both to be used, giving a combined capacity of 169,000 gal. Rails are laid in the sumps so that 2-ton Hudson cars may be run in to remove sludge.

Two Mather and Platt Plurovane pumps, powered by 150 hp a.c. motors, are installed in the pump pit below the pump room floor. Pump capacity is 500 gpm each at a head of 828 ft. Pumping is automatic as either or both pumps are controlled by float switches in the suction area between the dams.

In the topman's office on surface and in the 750 station are indicator lights and recording meters for both the main mine pumps on the 750 level and the 990 pumps.

Ventilation. Ventilation is provided by a 60-in. axivane fan with an adjustable aluminum blade, capable of 85,000 cfm at 3.65-in. water gage. The fan is on the 750 level at the foot of a 6x8-ft ventilation raise to the 1st sublevel. The fan draws contaminated air out of each slusher drift through a 2x4-ft opening, equipped with a butterfly regulator beside the tail post, down into the 4x7-ft ventilation drift running through the center of each mining block. Steel ventilation doors and bulkheads placed on each sublevel prevent any recirculation of air from the shaft. Fresh air is downcast in No. 1 shaft, through haulage ways and into the slusher drifts at the loading openings. Thus fresh air enters each slusher drift immediately in front of the slusher operator and moves along the drift to the tail post end, carrying the dust and smoke.

Concreting Plant

All units of the underground concreting plant are mounted to operate on 36-in. track gage and may be moved to any location on the main haulage level. The plant consists of the following mobile equipment plus the aggregate raise and batcher.

1—Three batch cars—these are 4-wheeled cars with ten separate compartments, each compartment holding the measured amount of sand and stone as weighed out by the batcher for ½ cu yd of concrete. Each car has its own discharge belt running full length of the cave under sliding discharge doors on the compartments.

2—Concrete mixer is a standard ½-cu yd mixer mounted on wheels and powered by a 10 hp 250 volt d.c. motor.

3—Concrete conveyor is a 16-degree conveyor, approximately 28 ft long, which conveys the mixed concrete from the discharge of the mixer to the feed hopper of a pumpcrete machine. The conveyor belt is powered by 250 volt d.c. motor.

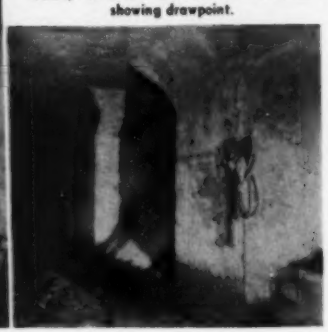
4—Pumpcrete machine—a standard 160-S pumpcrete machine powered by a 25 hp 250 volt d.c. motor, capable of pumping concrete through 6-in. pipe 160 ft vertically or 800 ft horizontally.

5—Six-in. pumpcrete pipe, including bends, gaskets, and couplers.

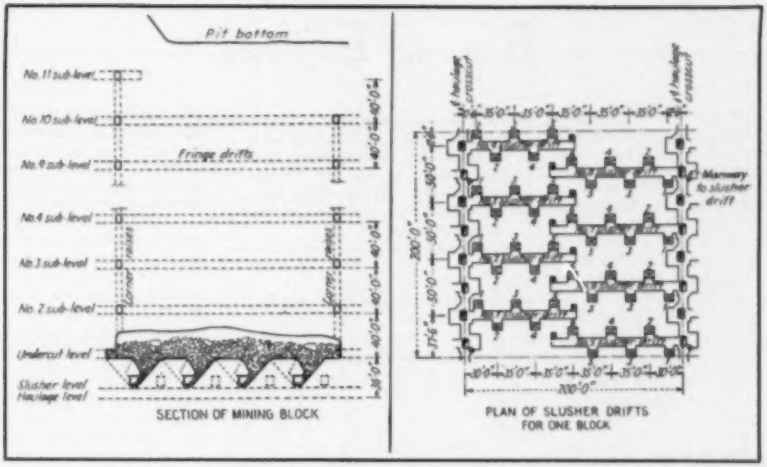
In addition to this equipment, which is a complete concrete mixing and placing plant, stand-by units, with motors, are on hand for the mixer, conveyor and pumpcrete machine.

The aggregate storage raise runs from the 700 level to the aggregate station near the tipple on the 750 level. It is concreted, with a steel plate lining and partition. The rock portion will hold 59 cu yd, and the sand portion 32 cu yd. Two-ton rocker dump cars are loaded by scoopmobile on surface, caged to the 700 level, trammed 135 ft and dumped into the raise. An automatic batcher is installed at the raise discharge in the back of the 750 level.

Up to the end of 1950 a total of 37,329 cu yd of concrete have been poured in the underground program. This includes 8826 cu yd poured in the headframe, surface bins, surge bin and shaft.



For left, underground crusher room. Note concrete arch. Center, aluminum forms set up in slusher drift, ready for concrete. Below, finished concrete slusher drift showing drawpoint.



Layout of 200x200-ft block and section showing the cutoff drifts that perimeter the block to form a cutoff for initiating the caving action. Slusher drifts are spaced 50 ft apart on the same side of the haulage drift, and alternate drifts are 25 ft apart.

Geology and History

Asbestos minerals are divided into two main classifications: the amphibole group and the serpentine group. The amphibole group, made up of silicates of magnesium, iron and calcium, are resistant to fire and acids, but are not the most desirable for spinning. Minerals of this group include anthophyllite, tremolite, actinolite, crocidolite, and amosite. Asbestos of the serpentine group is a hydrous silicate of magnesium; the fibrous variety is known as chrysotile asbestos. Chrysotile constitutes the largest proportion of world asbestos production, and has excellent spinning characteristics. This mineral is usually found in various shades of green, occasionally light yellow or cream; however, the separated fibers are white.

Serpentinization of peridotite produced the deposits at Asbestos, with the development of chrysotile as cross-fiber veins predominating, although slip fiber in veins and slip planes also occur. There is considerable fiber that is not in veins or slip planes, but disseminated through the serpentinized peridotite; however, there is little development of massive serpentine done in the mine. The fiberized orebody of Jeffrey mine, for simplicity, may be considered to be cylindrical in shape, tilted on its long axis and dipping from northwest to southeast. There is no regularity to the fiber occurrence, as barren and fiberized masses are found both horizontally and vertically throughout the mine. About 25 pct of the orebody is barren rock. On the north side the orebody is in contact with a slate footwall, striking north 60° east and dipping southeast at approximately 45°. On the east and west sides the ore fades irregularly into unaltered peridotite and noncommercial fibres. The vertical extent of the ore body has never been determined, boreholes have not indicated the depth of the ore. Drilling shows that ore extends more than 1000 ft below the surface and has a horizontal area of about 0.18 sq miles.

Erosion by glaciers and streams has removed considerable country rock and ore, especially along the south and east side of the orebody. Finely divided, highly waterbearing glacial drift has been deposited over this area.

Asbestos was first discovered in Quebec as early as 1862, but this discovery was not in any of the present producing areas. During 1876 asbestos was discovered at Thetford and mining proceeded rapidly. In the meantime, near Danville, a natural phenomenon that was puzzling the settlers of the district was masses of long silky fibers apparently growing within the rock and was locally known as "Webb's Ledge" since it occurred in an out-cropping on the farm of Charles Webb.

Even though asbestos had been recognized in Thetford only 60 odd road miles away, the phenomenon of Webb's Ledge remained a local curiosity until Evan Williams, a Welshman, while visiting his brother, recognized it. Williams tried unsuccessfully to interest local farmers in the out-crop, and eventually went to Richmond to try to interest W. H. Jeffrey. Jeffrey, an Englishman, apparently of considerable substance, opened the mine on a royalty basis in 1881. Thus the Jeffrey mine came into being, and the village of Asbestos took form and expanded with the mine.

Jeffrey operated the mine for about 13 years selling his product, which consisted solely of crude, to the H. W. Johns Manufacturing Co., in Manhattan, N. Y. In time, however, he failed to pay the royalties, and lost a lawsuit to Webb. He also became deeply indebted to his miners and creditors and went into bankruptcy. Shortly after, the Asbestos and Asbestic Co. was formed to develop the property and in 1897, the Asbestos and Danville Railway was built to connect with the Canadian Northern Railroad at Danville.

In 1905, T. F. Manville became interested in the Asbestos and Asbestic Co., and accumulated stock until he held the controlling interest. This company went into liquidation in 1916 and the property was taken over by the Manville Asbestos Co. Ltd., backed by American capital. In 1918 the company was reorganized under the name of the Canadian Johns-Manville Co. Ltd.

Books for Engineers

Proceedings of the Fourth Empire Mining and Metallurgical Congress, Great Britain 1949. Part II published at the Offices of the Congress in London, England, 1139 pp. The price of these proceedings is £2. The parts may be purchased separately at a cost of £1 each.—Part I of this book covers a report of the organizing committee, the inaugural meeting and presidential address, banquet at Guildhall, welcome by Sir Richard Livingstone at Oxford and final session. Technical session and papers are also covered in Part I. Part II covers such topics as coal, present-day trends in mineral dressing and metallurgy.

Mining Engineer's Survey Manual. By J. E. Metcalfe. Published for "Mine & Quarry Engineering" by Electrical Press, London, 1951. 341 pp., illus., charts, diagrs., tables, 8¾x5½ in., cloth, 30s., available from Percival Marshall & Co., Ltd., London, and British Book Centre, 122 East 55th St., New York 22, N. Y.—Beginning with a brief treatment of surveying fundamentals and a discussion of the correlation of surface and underground surveys, this practical manual continues with descriptions of surveying practice both above and below ground including stopes, inclined shafts, etc. Space is devoted to sampling and valuation, geological considerations, and descriptions of modern surveying instruments. The organization of survey operations is covered in the last chapter.

Tables for Microscopic Identification of Ore Minerals. By W. Uytenbogaardt. Princeton University Press, Princeton, N. J., 1951. 242 pp., tables, 10¼x7¼ in., cloth. $5.—The tables in this practical laboratory handbook deal with all ore minerals which have been described sufficiently to permit identification. The tabulated properties are name, chemical composition, crystal system, Talmage hardness, color aspects, etch tests, and a variety of miscellaneous recognition characteristics. The primary arrangement is generally in order of increasing polishing hardness, although ore minerals which are closely related or commonly occur together are grouped together.

Alkali Soils. By W. P. Kelley. Reinhold Publishing Corp., New York, 1951. 176 pp., illus., charts, tables, 9¼x6 in., cloth, $5.—Beginning with the origin and accumulation of salts in soils, the author goes on to discuss chemical, physical and microbiological effects and the determination of soluble salts and exchangeable bases. The practical aspects of the book are dealt with in the chapters on irrigation in relation to alkali soils, on alkali soils in relation to plant growth, and on the reclamation of alkali soils.

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Mathematics for Engineers. By R. W. Dull, rev. & ed. by R. Dull. 3 ed. McGraw-Hill Book Co., New York, Toronto, London, 1951. 822 pp., diagrs., charts, tables, 8¼x5½ in., cloth, $7.50.—Beginning with algebra and continuing through logarithms, determinants, the slide rule, trigonometric functions, the calculus, and dimensional and similarity analysis, this work affords a convenient review of those phases of mathematics which are especially important in engineering work. It is intended either for use as a practical reference work or as a text for private study. A chapter on differential equations has been added, and other revisions have been made to bring the book up to date.

Soil Mechanics, Foundations, and Earth Structures. By G. P. Tschebotarioff. McGraw-Hill Book Co., New York, Toronto, London, 1951. 655 pp., illus., diagrs., charts, tables, 9½x6¼ in., cloth, $6.50.—Introductory without being elementary, this text covers theories of soil mechanics and the related design and construction practices of soil and foundation engineering. Emphasis is placed on experimental data and field observations illustrated by examples of both successful and unsuccessful structures. The book is intended for use at undergraduate and graduate levels and as a general reference. Special topics include the triaxial cell tests, allied geotechnical sciences (geology and soil physics), and soil engineering problems in permafrost regions. Extensive references are included to facilitate further study.

Compressed Air in Mining and Industry. Compiled and edited by S. H. North. Sir Isaac Pitman & Sons, Ltd., London, 1951. 170 pp., illus., diagrs., charts, tables, 8¾x5½ in., linen, 18s.—Compiled largely from material provided by experienced compressed-air engineers, this book covers a wide field of applications and gives sufficient data to indicate the advantages and adaptability of this form of power. The most suitable plant and equipment required for carrying out any particular work—in mining, quarrying, or industry is described.

The Seventh Edition of Volume II of The System of Mineralogy. Published by John Wiley & Sons, 1124 pp. $15.—Covering halides, nitrates, borates, carbonates, sulphates, phosphates, arsenates, tungstates, molybdates, etc., the new Volume II offers a complete description of each mineral included. As in the seventh edition of Volume I, published in 1944, mineral descriptions in Volume II consist of the following items of information: Classification, morphological and X-ray crystallography, physical properties, optical properties, chemistry, occurrences, name derivation, synonomy, and bibliographic references. Revisors Charles Palache, the late Harry Berman, and Clifford Frondel, all of Harvard University, have continued in the new edition of Volume II the changes introduced in the revision of Volume I. These changes include a new mineral classification based on crystal chemistry; a new elastic series of classification numbers for species; revised morphological elements based on the structural unit cell; revision of specific gravities based on new observations, etc.

Manhood of Industrial Engineering by K. T. Korn, has just been published by Edward A. Berk & Associates. In his book, Korn reviews the history and development of industrial engineering, and its growth into a profession. His extensive research indicates that the science of management was conceived simultaneously with, and as a part of, the industrial revolution. The author goes on to develop the evolution of scientific management from the early formulation, around 1900, through the period of the efficiency expert of the irresponsible twenties to the growth of industrial engineering into a dignified profession. The author shows an actual application of the principles of his infinite-valued orientation in a case history. Here he illustrates his methodological approach to a specific, typical industrial problem, and points out the amazing results which can be achieved, particularly in creating lasting harmony in labor-management relations for the benefit of all: Industry, worker, management, consumer, investor, etc.

The Seamless Story. By J. P. Boors. Commonwealth Press, Los Angeles, Calif., 1951. 285 pp., illus., tables, 9¼x6 in., fabrikoid, $5.75.—A non-technical history of seamless tubing, this book traces the evolution of tubing without seams or welds from its invention to contemporary mills. Biographies of inventors and important persons in the industry, and information about companies are given. Included is an extensive bibliography listing patents, technical articles.

Pipeline Transportation of Phosphate

by I. S. Tillotson, R. B. Burt, and James A. Barr

THE pumping of solids in water suspension is an important part of many metallurgical and mining operations. In most cases, it is still in the rule of thumb category for which no universal formula has been developed, and much research is needed.

Because of the limited and incomplete data available, this article may be classed as an experience paper, which is presented with the hope that some contribution will be made toward the development of the so-called universal formula. This formula, if and when developed, may be evolved from several factors, many of which are not now available for general application.

The designing engineer is interested in obtaining accurate forecasts on: 1—the minimum velocities needed to prevent choke-ups in the pipeline, which in turn dictates pipe sizes, 2—power required for pumping, 3—pump selection. The basic factors for a given problem will include: 1—weight per unit of time of solids to be handled, 2—specific gravity of solids, for calculation of volume, friction and power, 3—screen analysis of solids with the colloidal acting, i.e., the slime fraction, a very important factor, 4—shape of particle or some means of determining a friction constant, 5—effects of percentage of solids, 6—development of a viscosity factor to be used in the overall calculations, 7—calculation of the lower limits of pipeline velocities permissible, 8—calculation of total head, pump horsepower, and 9—setting up of pump specifications.

In certain limited cases horsepower and total heads and minimum velocities may be computed and a suitable pump selected from basic data, but in many cases, as in mining of Florida pebble phosphate, experience rather than a hydraulic formula still should be used as a basis of selection.

Pumping Florida Pebble Matrix

Pumping at the Noralyn mine of International Minerals and Chemical Corp. will be used as an example. Other areas will vary as to the characteristics of the matrix, especially the slime content.

A typical screen analysis of this matrix is: +14 mesh, pebble size,* 2.1 pct; −14 +35 mesh, 11.4 pct; −35 +150 mesh, 60.5 pct; −150 mesh, 25.0; total, 100 pct; moisture in bank, 20.0 pct; weight per cu ft in bank, 120 lb.

The −150 mesh fraction may increase to as much as 35 pct in adjacent areas.

When thoroughly elutriated, the matrix has a relatively slow settling rate, which is an important factor in permitting lower pipeline velocities without choke-ups. Exact data is not available to evaluate settling rates. For a factor of 100 a suspension of clean building sand in water is suggested. When pumping long distances, a quick settling matrix allows the coarser solids to settle out along the bottom of the pipeline, causing drag, turbulence, and increased friction. With a slow settling matrix as at Noralyn, turbulence acts to keep the solids in suspension at a lower friction head, regardless of the pumping distance.

When the pebble content of the matrix, i.e., the +14 mesh fraction, is in excess of 10 pct of the total solids, trouble may be expected from settling out even in normal pumping distances. To prevent choke-ups and maintain tonnage, an additional pump must be added in the long runs, where one pump would otherwise be satisfactory.

A typical pulp handled is: total volume, 7800 gpm; water, 4500; solids pumped per hr, 4200 lb; sp gr pulp, 1.4; percent solids in pulp, 40.; pipe size, 16-in. ID; pulp velocity, 12.85 fps; probable critical velocity, 10 fps, as below this minimum velocity choke-ups would be numerous.

In calculating friction heads the Armco handbook is used where a roughness factor based on 15-year-old pipe is set up.

Because the pipe used in pumping matrix is smooth and polished because of the scouring action of the phosphate and its silica content, the head losses in the Armco table for water are practically the same as in pumping the Noralyn matrix through smooth pipe, plus the fact that conditions vary widely over short periods, making accurate determinations difficult to obtain.

New pumps and pump changes are being tested continuously and a wealth of data built up. This has resulted in a substantial improvement and lower relative costs in pumping matrix. The Florida phosphate industry is constantly seeking to offset higher wage and material costs with improved technique.

Until a few years ago a 12-in. discharge pump was commonly used, with heads as low as 80 ft. Sizes have gradually increased and heads more than doubled.

For example, the following pump was placed under test at the Noralyn mine: make, Georgia Iron Works; size, suction 16 in., discharge 14 in.; impeller, 39-in. diam; motor, 600 hp, slip ring; full load speed, 514 rpm.

The results were increased head, higher capacity than the older design, with fewer pumps in the line from mine to washer.

* Pebble is a commercial designation for the coarser fraction of finished phosphate from a washer, usually +14 mesh.

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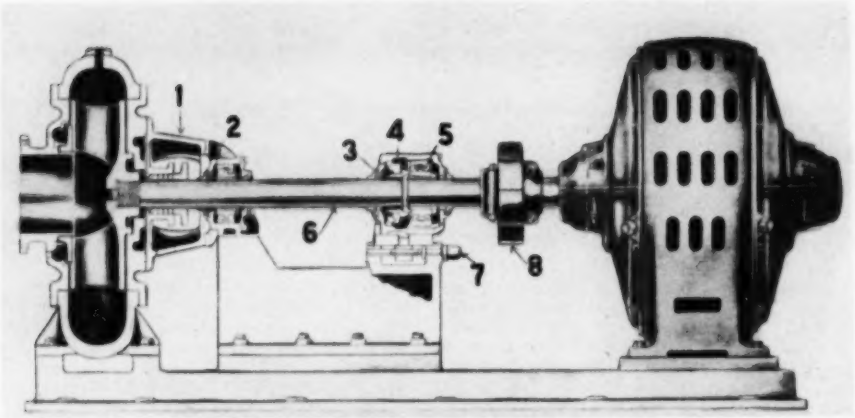

Fig. 1—General design of the discharge pump.

1—Rigid construction to support pump shell and maintain alignment.
2—Double Klosure on forward end with flinger retains oil and prevents entrance of water.
3—Labyrinth seals in combination with flinger effectively retains oil and guards against contamination.
4—Steep angle SKF spherical roller thrust bearing.
5—Radial SKF spherical roller bearing arranged to carry all radial load without thrust and self aligning with thrust bearing.
6—Heavy alloy steel shaft.
7—Precision adjustment for impeller clearance 2-in. screw with 12 threads per in.
8—Heavy duty flexible coupling built by G.I.W. especially for severe service.

The next pump tried was: size, suction 16 in., discharge, 14 in.; impeller, 43¾-in. diam. A typical test indicated: pulp capacity, 7100 gpm; solids, 625 cu yd bank measurement per hr; head, 160 ft (water equivalent); water added, 4580 gpm; brake hp, 490; pump efficiency, 78 pct. Level pumping distance, one pump with 16-in. pipe 7000 ft as against 3000 ft with the 12-in. pump formerly used.

The general design of the pump is shown in Fig. 1 and the operating characteristics of the 16-in. size are shown by the curves, see Fig. 2.

The latest pump tested is: size, suction and discharge 16 in.; impeller diameter, 49 in.; and designed for: efficiency, 65 pct; capacity, 6700 gpm; head, 188 ft; brake hp pump, at 505 rpm, 650.

The former pump did not fully load the motor because the efficiency of the pump was greater than design calculations indicated.

Pumping distances, head, volume, and tonnages vary according to plant demands and operating conditions; pump speeds must be varied frequently, often below peak efficiencies. So far, the slip ring motor has been the only practical drive found, but consideration is being given to other types of ac motors, hydraulic or magnetic couplings, and dc motors served by rectifiers.

Formerly 600-rpm motor drives were used, but for the existing conditions it was found that 500 rpm gave smoother operation at higher efficiency.

In the course of test work, one formula has evolved which has been found suitable for calculating runner diameters, i.e., $D = \frac{1837 \cdot \sqrt{H(0.93)}}{\text{rpm}}$, where D is in inches, and H = water equivalent of head, in feet.

At the Georgia Iron Works, M. R. Creasy, Chief Engineer, equipped a trailer for field test work, as shown in Figs. 3 and 4, for obtaining the necessary test data. Velocity of flow is determined by the electric titration method at 20-min intervals.

Table I. Data and Results, Achan Mine, International Minerals and Chemical Corp., G.I.W.,* 39¼x12 In., 16-In. Suction, 38-In. Diam

Matrix Line Velocity, Ft per Sec	Gpm	Suction Head Equiv. Ft Water	Discharge Head, Equiv. Ft Water	Total Dynamic Head, Equiv. Ft Water	Water Hp	Kw	Rpm	Brake Hp	Efficiency, Pct
12.69	6104	12.00	187.11	199.11	306.90	360	580	439.14	69.80
13.33	6412	12.46	187.11	199.57	323.14	372	580	453.80	71.20
11.56	5575	12.00	187.11	199.11	280.31	360	580	439.14	63.80
12.31	5922	10.20	187.11	197.31	295	360	580	439.14	67.10
13.56	6523	11.90	187.11	199.01	327.81	390	580	475.7	68.90
13.56	6523	16.54	184.80	201.34	331.65	390	580	475.7	69.70
11.59	5575	9.06	189.42	198.48	279.43	360	580	439.14	63.60
13.11	6306	17.00	191.73	208.73	332.39	372	580	453.80	73.20
13.33	6412	11.33	189.42	200.75	325.05	372	580	453.80	71.60
11.94	5744	15.86	182.49	198.35	287.71	360	580	439.14	65.60
Avg 12.70	6109	12.84	187.34	200.18	308.93	369.60	580	450.85	68.44

* Georgia Iron Works.

A tabulation of the test of a 39¼x16x12-in. pump is given in Table I. Data on matrix handled was gathered by the Engineering Department of International. Some of the previous records are of value in making comparisons; these are shown in Tables II and III. The effects of partial speeds and worn pumps apply to old as well as the later design pumps.

Fig. 5 shows the general exterior design of an Allen-Sherman Hoff G frame Hydroseal pump, now used in the Florida phosphate field. The clear water characteristics are given in Table IV.[1]

Pumping Tennessee Matrix†

The Tennessee Valley Authority built and operated a 3-mile pipeline between the Akin mine and the Godwin beneficiation plant, in Maury County, after a thorough study of various forms of transportation. Because of the terrain and operating requirements, pipeline transportation has proved the most economical and has furnished valuable hydraulic data.

The matrix, consisting of a conglomerate mass of clay, silica, and phosphate particles, is mined from open pits by draglines at the rate of 70 to 90 tons per hr.

The matrix next is crushed in a hammer mill, to −1½-in. size, slurried with water, pumped to a blunging-classifying system for desliming at a −10 micron split, which overflows the final thickener into tailing ponds. The solids thus wasted consist principally of clay.

† Abstracted from "TVA's Experience with Hydraulic Transportation of Phosphate Concentrates," by R. B. Burt.

Table II. Data on Pumping from Parker Property to No. 92 Washer, Operated from March 26 to 30, 1938

	"F" Frame ASH* Lift Pump, 39-In. Runner and "E" Frame ASH Pit Pump 32-In. Runner
Time operated, hr	53.5
Matrix pumped, cu yd	13,203
Matrix per hr, cu yd	246.7
Rock mined, tons	3,312
Rock per hr, tons	61.9
Matrix per ton rock, cu yd	3.98
Water pumped, gpm	4,400
Solids pumped, pct	25.32
Distance from lift pump to washer, ft	3,080
Distance from lift pump to pit pump, ft	1,220
Total pumping distance, ft	4,300
Size of pipeline, in.	14
Static head on lift pump, ft	−5.5
Total static head, ft	36.02
Suction head on pit pump, ft	22.6
Discharge pressure on pit pump, lb	52
Suction pressure on lift pump, lb	+4
Discharge pressure on lift pump, lb	92
Total head on pit pump, ft	142.7
Total head on lift pump, ft	205.3
Total dynamic head, ft	346.0
Friction head per 100-ft pipe	7.21
Kwh used	34,880
Kwh per hr	651.9
Kwh per cu yd matrix	2.642
Water hp	514.8
Electrical hp	573.8
Motor efficiency, pct	90
Overall pumping efficiency, pct	65.46
Saving in favor of ASH* pumps, pct	10.6

* Allen-Sherman Hoff.

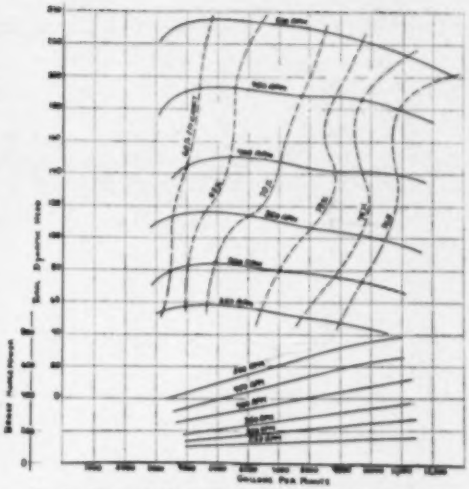

Fig. 2—Operating characteristics 16x52-in. dredge pump, 51 in. diam impeller, 16 in. diam suction.

The 1½ to 65 mesh and the 65 mesh −10 micron splits called concentrate are mixed in a surge tank, which is brought to the proper pumping density with water for pumping through the pipeline.

There is little reserve storage, and the pumping is continuous to the Godwin plant where the −10 micron size left from the first desliming or formed by abrasion in the pipeline is separated again and sent to tailing ponds.

The deslimed sands, often called washed sands, are drained in wet storage piles and shipped to the TVA electric furnace phosphorus plant about 80 miles away at Wilson Dam, Ala.

Because of the engineering value the following section is presented in its entirety.

Hydraulic Transportation System

In general the design problem presented by the requirements of the flowsheet of the beneficiation

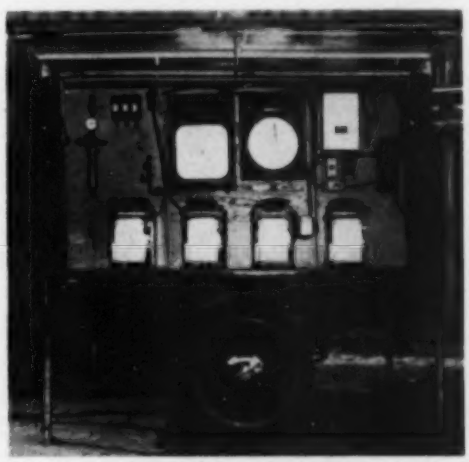

Fig. 3—Trailer equipped for field test work.

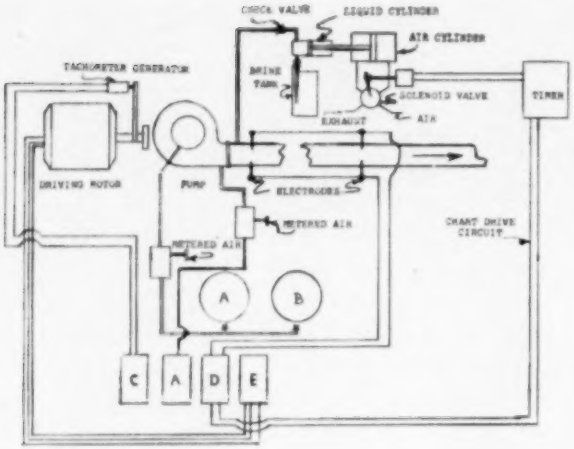

Fig. 4—Schematic diagram for testing phosphate pulp line pumps with Georgia Iron Works Co. testing unit.

A—Recording pressure gage.
B—Recording vacuum gage.
C—Tachometer.
D—Conductivity recorder.
E—Recording kw meter.

plant was to provide a hydraulic transportation system consisting of pumping equipment and accessories and a pipeline approximately 16,000 ft long for transporting up to 90 tons per hr of concentrate, as a water slurry or pulp, from the Akin plant to the Godwin plant.

Since the preceding operations of mining, scrubbing, and concentrating were to be successive and continuous, provision was made to insure uninterrupted operation of the concentrate line. Other factors which influenced the selection and location of pumping equipment and the determination of pipe diameter were: 1—the water requirement of the classification equipment at Godwin, 2—the pumping pressures required, and 3—the lowest velocity at which the required tonnage of sand and water could be delivered without deposition of solids in the pipelines.

Since the water used to transport the solids was also to serve as dilution water in the classifiers at Godwin, it was desirable that the quantity be held within limits necessary for proper classification with the addition of a minimum of makeup water at the Godwin plant. This resulted in a flow requirement of from 1200 to 1500 gpm.

Safe operating pressure on pumps suitable for the system imposed a further limitation on design. In selecting pumping equipment, a choice was made between the use of five low-head pumps, or three comparatively high-head pumps. It was necessary that the pumping layout require no pumping pressures in excess of 150 psi. It was anticipated that with a high-head installation, pump maintenance costs would be greater and more frequent renewal of pump parts would be necessary than with a low-head installation. However, based on estimates of installation, operating, and maintenance costs, the high-head installation appeared to be more economical.

Theoretical Considerations

Previous investigators have reported that two types of flow exist for slurries or pulps similar to the laminar and turbulent states of flow which are characteristic of true liquids.[2-4] Also, it has been found that Poiseulles' equation for viscosity does not apply to laminar flow of slurries as it does for true liquids, although it does apply to both type fluids in the turbulent flow range, and in this range the viscosity of the slurry is the same as that of the liquid in which the solids are suspended.[2] It has been found that with slurries, laminar flow changes to turbulent flow at a rather clearly defined velocity, referred to as the critical velocity, and that this velocity depends on the solids content of the slurry and the particle size distribution of the solids.[5, 3] Work of other investigators indicated that in pumping sludges, pulps, and muds, the Fanning friction factor, f is proportional to the Reynolds number; that the friction factor varies with velocity to about the same extent in slurries as in true liquids within the turbulent flow range, and the factors for slurries are only slightly higher than those for true liquids. The latter difference also depends on the solids content of the slurry and the particle size distribution of the solids.[5, 6]

Although the mechanism of transporting slurries in pipelines had been investigated considerably in small scale tests, with limited verification by data on dredging operations, most of the information reported involved material dissimilar to phosphate

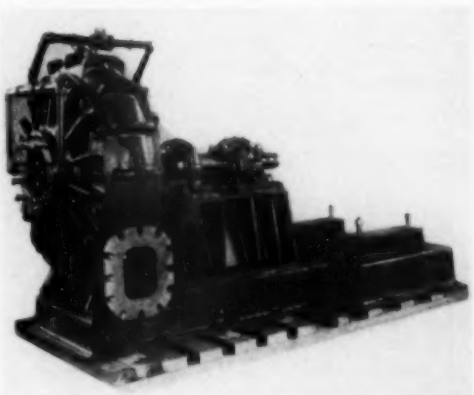

Fig. 5—G-frame hydroseal dredge pump, height, 6 ft, 5 in.; length, including base, 12 ft, 4 in.; weight, 19,000 lb.

Table III. Pump Tests, July 28, 1941. Type 711 Pump Tested, 12-In. G.I.W., HD 39¼-In. Matrix Pump, Mine No. 122[a]

4½-In. Calibrated Nozzle								
Controller position	Switch	2	4	6	8	10	12	Open
Rpm	400	416	428	438	320	338	360	382
Vacuum, in.	7½	7¾	8	8	9	9¼	10	10
Nozzle pressure, lb	29¼	31	33	38	48	53	58½	63
Total dynamic head, ft	76.6	80.1	85.3	96.8	121.1	130.9	141.8	156.9
Gpm	2980	3030	3130	3350	3770	3920	4080	4320
Water hp	87.3	61.4	67.5	81.9	113.2	128.5	148.5	171.0
Motor load, kw	123.2	128.0	134.0	152.0	194.0	206.5	224.0	252.0
Electrical hp	165.0	171.5	179.4	203.5	260.0	276.0	303.0	338.0
Overall efficiency	34.7	35.8	37.6	40.2	44.3	47.0	48.4	50.7
Pump efficiency[b]								56.3
5½-In. Calibrated Nozzle								
Controller position	Switch	2	4	6	8	10	12	Open
Rpm	400	412	422	440	500	535	552	580
Vacuum, in.	8½	9	9	9¾	11	11¾	12¾	14
Nozzle pressure, lb	29½	25	26½	31	42	46	51	57½
Total dynamic head, ft	63.9	67.9	71.4	82.4	109.5	119.3	132.0	148.7
Gpm	3930	4060	4180	4320	5200	5500	5780	6150
Water hp	63.5	69.5	75.5	94.0	143.5	165.7	192.7	230.5
Motor load, kw	134.8	141.2	146.0	173.2	228.0	248.0	270.8	306.0
Electrical hp	180.5	189.0	195.5	232.0	305.0	330.0	363.0	410.0
Overall efficiency	35.2	36.8	38.6	40.5	41.6	50.2	53.2	56.3
Pump efficiency[b]								62.5

[a] Service prior to test: pump runner, 1053½ hr; pump shell, 1053½ hr.
[b] Assuming motor is 90 pct efficient at full speed.

concentrates, and it was not believed that these data were adequate for the design of TVA's transportation pipeline.[4-6] Therefore, to study more thoroughly the effect of slurry properties on transporting characteristics and to provide data for design purposes, an experimental program was carried out on pumping phosphate slurries containing from 2 to 35 pct solids through 400 ft of 2 and 4-in. diam iron pipe at velocities of from 7 to 11 fps. These velocities were found to be above the critical velocity for this system; below a velocity of about 7 fps the solids tended to settle out and resulted in stoppage of the pipelines.

In this work it was observed that the pumping of the phosphate concentrates through the pipe polished its walls to such an extent that the head loss due to friction was lower than has been reported as normal for steel pipe. Specifically, the surface smoothness constant, C, used in the following Williams and Hazen formula was 160, whereas, this factor is normally 140 to 150 for steel pipe.[7] On the basis of this, friction factors in the lower range were used in determining the requirements for the plant pipeline.

$$Hw = \frac{(V)}{(1.318\ CRO.63)}\ 1.85$$

Where: Hw = friction loss, in feet of water per foot of pipe length; V = velocity, in feet per second; C = constant, varying with the kind of pipe and the conditions of the wall; and R = the hydraulic radius.

In transporting slurries or pulps through a pipeline of given diameter the effect of variation in the solids content of the pulp on the head loss while pumping at a given velocity, relative to the head loss resulting when water alone is pumped at the same velocity can be represented by the formula:

$$\frac{\Delta Hp}{\Delta Hw} = Y$$

Where, the particle size distribution of the solids is constant Hp = head loss in feet of pulp per foot of pipe length and Hw = head loss in feet of water per foot of pipe length.

It has been found that at values above the critical velocity the friction factor for slurries increases with a decrease in velocity and increases with an increase in pulp density (solids concentration).[8] Since head loss is directly related to the friction in pipelines, the ratio $\frac{\Delta Hp}{\Delta Hw}$ may be considered to increase with a decrease in velocity or an increase in pulp density

Table IV. Clear Water Performance of "G" Frame, Hydroseal Dredge Pump, 14-In. Suction, 12-In. or 14-In. Discharge

	435 Rpm				500 Rpm						585 Rpm						
U.S. Gpm	36-In. TDH[a]	Imp. Hp	37-In. TDH[a]	Imp. Hp	38½-In. TDH[a]	Imp. Hp	36-In. TDH[a]	Imp. Hp	37-In. TDH[a]	Imp. Hp	38½-In. TDH[a]	Imp. Hp	36-In. TDH[a]	Imp. Hp	37-In. TDH[a]	Imp. Hp	38½-In. TDH[a]	Imp. Hp	Imperial Gpm
4000	91	150	97	162	107	180	123	210	130	226	142	254	170	323	180	348	196	392	3330
4500	90	161	96	173	106	191	121	225	129	242	141	269	169	342	179	367	195	403	3750
5000	89	171	94	183	104	202	120	240	127	257	140	284	168	360	178	386	194	423	4160
5500	87	182	93	194	102	214	118	254	125	272	139	298	166	378	176	404	193	443	4580
6000	85	193	91	205	100	226	116	268	123	287	138	312	164	395	175	421	191	462	5000
6500	83	202	89	217	98	238	115	281	121	301	134	327	162	412	173	438	189	483	5410
7000	82	211	87	228	96	250	113	294	120	315	132	342	160	429	170	455	187	503	5830
7500	80	222	85	239	94	261	110	307	117	329	130	357	158	445	168	474	185	523	6250
8000	78	232	83	249	91	271	108	319	115	343	127	372	156	461	166	492	182	543	6660
8500	75	241	80	258	88	281	106	331	113	356	125	387	153	478	164	510	180	562	7080
9000	73	250	78	268	86	291	103	343	110	369	122	401	151	495	161	529	177	580	7496
9500	70	258	76	275	83	301	101	355	108	381	119	415	148	512	158	547	174	600	7916
10000	68	266	73	284	80	310	98	367	105	393	116	428	146	528	156	563	171	617	8330
10500	65	274	71	292	78	320	95	379	102	404	113	441	143	545	153	583	167	636	8740
11000	63	281	68	300	74	329	93	390	100	415	109	453	140	561	150	600	164	654	9160
11500	60	288	65	309	70	337	90	400	97	427	106	466	137	578	147	618	160	673	9580
12000	57	295	62	317	67	344	86	410	93	439	102	478	133	594	143	635	156	693	9990

[a] Total dynamic head.

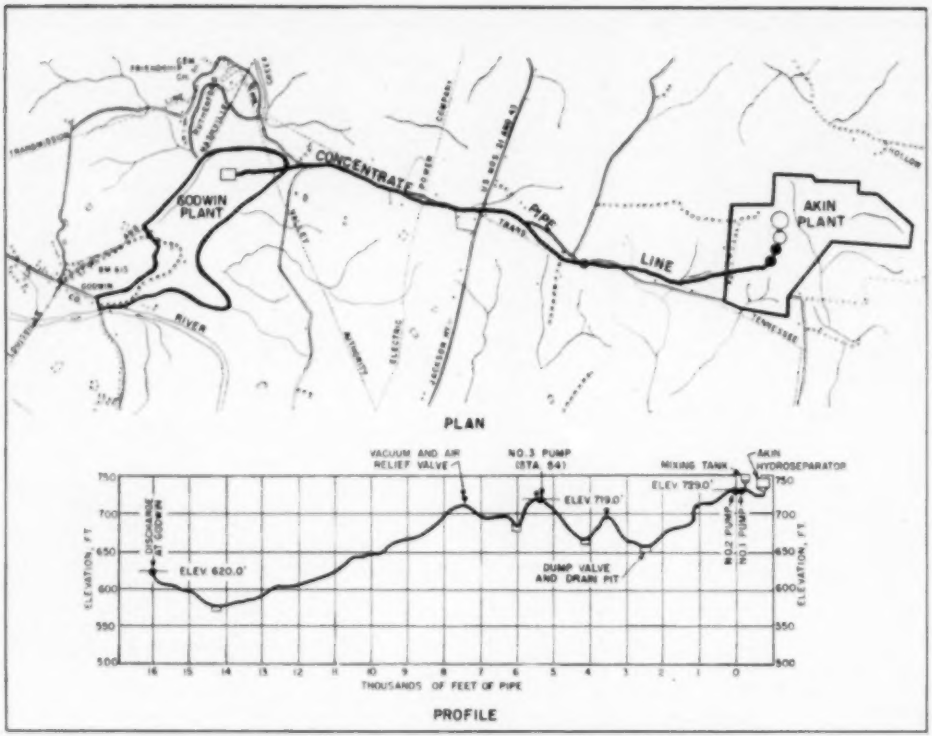

Fig. 6—Phosphate concentrate pipeline from Akin plant to Godwin plant.

and approaches a value of one at high velocities or very low pulp densities.

The ratio of $\frac{\Delta Hp}{\Delta Hw}$ calculated from experimental data averaged 1.5 at velocities of 7 to 13 fps when the slurry contained 25 pct solids. On this basis the desired transporting capacity of the pipeline, the total head loss, and pump requirements for pipelines of several different diameters were calculated. It was decided that with a minimum velocity of 8 fps (considered safely above the critical velocity), an 8-in. diam pipe would provide the necessary capacity with a maximum pulp concentration of 25 pct solids and would have pumping requirements within the range of standard pumping equipment.

Pipeline Location

After the size of the pipe diameter had been determined, it was necessary to make a detailed study of the location of the line and to determine the total head, select and locate the pumping equipment in the system, provide facilities to insure continuous operation, and incorporate certain features considered necessary for safe operation of the line.

From study of several preliminary locations, the final location shown in Fig. 6 was selected because it appeared to be the most practical route between the Akin and Godwin plants when consideration was given to distance, topography, and construction costs.

The line as originally installed was an 8-in., wire-wound, wood-stave pipe. Several test sections of cast-iron pipe lined with wood staves and sections of abrasion-resistant, spiral-welded, steel pipe were included in the line. After several months' operation, examination of the line revealed that the unlined steel pipe was withstanding the abrasive action of the phosphate sands better than the wood pipe, and experience has shown that maintenance of the wood pipe was high because of a leakage which required frequent detailed examination of the line. Therefore plans were made for conversion to an all-steel line as replacements became necessary. Subsequent experience has indicated that wear of the steel pipe amounts to only about 0.02 in. per year.

The 8-in. spiral-welded, abrasion-resistant, steel pipeline is laid in 40-ft lengths connected by flanges and Style 38 Dresser couplings. The couplings are used every third length to provide for expansion and contraction, and the play in the couplings, together with the flexibility of the pipe itself, permits long curves, which facilitate laying the line in the rolling topography of the area. The line is laid on uniform grades conforming closely to the surface of the ground. Small cuts and fills in both earth and rock were made where necessary to obtain uniform grades and vertical curves. At points where fill would have been excessive, and at points where the line crossed natural drainage channels, the pipe is laid on wooden trestles built on concrete foundations. At road crossings the line is buried. At about station 85 + 00 the line passes underneath U. S. Highway No. 31, the main highway from Columbia to Nashville, Tenn. At this point a 36-in. diam corrugated steel pipe was

jacked through the base of the highway fill and the concentrate line laid through this pipe. At station 141 + 50, the line crosses Rutherford Creek, a stream 85 ft wide. To support the line over this stream, a suspension bridge was built with a central span between towers of 120 ft and with 60-ft side spans between each tower and the anchorage. The main cables and the floor system girders are on 5-ft 8-in. centers that provide ample space for maintenance and a walkway for inspection.

Pumping Equipment

Three pumps are used, see Fig. 6. Pumps Nos. 1 and 2 are located at the Akin plant near the hydroseparator, and pump No. 3 is located on the line at station 54 + 00. All three pumps are 6-in., centrifugal, closed impeller sand pumps, and at a speed of 1200 rpm develop 265 ft of head of a pulp of 1.20 sp gr. No. 1 pump is directly connected to a 250-hp, 1200-rpm, 2300-v synchronous motor running at a constant speed under a fairly constant load for power factor correction. Pump No. 2 is driven by a 250-hp, 2300-v, slip ring motor with speed variations from 640 to 1170 rpm and develops heads from 80 to 250 ft. No. 3 pump is also driven by a 250-hp, 2300-v, slip ring motor having speed variations from 760 to 1170 rpm and develops heads from 110 to 250 ft. The piping is arranged so that any combination of pumps can be used to transport the concentrate. The sand pumps are provided with water seals to prevent the passage of sand from the pumps into the bearings. A supply of clean, high-pressure, water is provided for the seals by small low-capacity, centrifugal pumps.

It was anticipated that the combinations normally used would be pumps Nos. 1 and 3 or with pumps Nos. 2 and 3, and calculations indicated that these combinations would result in a velocity in the line of from 8 to 9 fps, a flow of from about 1285 to 1450 gpm, pump pressures from 140 to 170 psi, and delivery rates of 95 to 110 tons of solids per hr when the pulp contained 25 pct of solids. The three-pump installation provides considerable flexibility in the total head that can be placed on the line. The No. 2 pump is considered a standby for the No. 1 pump; it also can be thrown on the line when No. 3 pump is down, and in cases of emergency due to surges or a too heavily loaded line, when it is necessary to use all three pumps. It has been found that with close control the use of three pumps is not frequently necessary.

When transporting 80 tons of concentrate per hr at velocities of from 8 to 9 fps, 40 to 45 tons of solids are in the line. In case of power failure, or if for any other reason all pumps are down and the problem of accelerating the mass in the line when starting up again is encountered, there is danger of the line becoming completely plugged. As a safety measure, and to provide means for getting the line back into operation without excessive loss of time, dump valves and vacuum and air relief valves are located at four points in the line as shown in Fig. 6. A drain pit with sufficient capacity to hold the pulp contained in the line between the two high points is provided under each drain valve. The pits are constructed with drains near the bottom that permit the water to drain out of the sand, which then may be reclaimed from the pits with trucks. The dump valves were designed originally for automatic operation, being held closed by an energized solenoid so that in case of power failure the solenoids would become deenergized and the holding mechanism released, allowing the valves to dump. In actual practice, however, the automatic operation of the valves was not entirely successful so they are now manually operated, although dumping is not frequently necessary.

To insure continuous operation, the pumps are provided with a dual power supply consisting of two substations feeding from a single 12-kv, three-phase transmission line. The two substations, one at the Akin plant and the other at station 54 + 00 on the concentrate line, have 2300-v secondaries. The pumps can be fed from either substation or from both substations operating in parallel. When the substations are operating in parallel, the transformers at each station are protected by directional overcurrent relays. In the case of a transformer failure at either station, the entire transformer bank at that station is automatically cut out of the circuit, which allows the concentrate pumps to continue feeding from the remaining transformer bank.

The flow rate in the line is determined by means of a velocity meter at the discharge end of the line, which consists of a 6-in. diameter nozzle and a scale calibrated to read in gallons per minute. The flow rate is determined by noting the intersection of the trajectory of the stream on the calibrated scale located at a fixed distance below the center line of the nozzle.

The discharge end of the line, the pump at station 54 + 00, and the two pumps at the Akin plant are connected by an automatic magneto monophone system for interplant communication. When the flow deviates too greatly from the desired rate, the pump operators are notified by telephone, and the pump speeds are adjusted accordingly.

Hydraulic Transportation System

After the wood pipe had been replaced with steel pipe, a test was made of the hydraulic transportation system, while operating under normal conditions during which only pumps Nos. 1 and 2 were used. Pressure gages were installed at all the principal points in the line, recording wattmeters were connected to each of the concentrate pump motors, and the speed settings of the rheostats on the variable-speed motors were calibrated. The pulp was sampled at the discharge end of the pipeline to determine the solids content and the particle size distribution of the solids. The data collected at 30-min intervals were averaged for periods of approximately 8 hr during which there was little variation in the operation. Data were also collected for two short periods on part of the line, station 54 + 00 to Godwin, while pumping water only, to provide a comparison with the data on slurries. The results of these tests are given in Table V. These data show that the load on the No. 1 pump motor was practically constant, and the additional pumping load required was taken up by pump No. 2 as was anticipated. The speed-head-power relationships of pumps Nos. 2 and 3 are shown in Figs. 7 and 8. Normal operation during the test period involved the use of pumps Nos. 1 and 2 only, but it is believed that the consumption of energy for pumps Nos. 1 and 3 would be about the same as that for pumps Nos. 1 and 2. Increasing the solids content of the pulp from 11.1 to 17.4 pct resulted in a decrease in the energy consumption from about 4.6 to 3.0 kw-hr per tons of dry solids transported, see Fig. 9. It may be also noted from Fig. 9 that the power

consumed per ton of dry solids pumped decreases with an increase in the solids transportation rate. About 70 pct of the energy consumed in pumping was used by pump No. 1. During one 8-hr period, period No. 8, pump No. 2 was used alone, and the power consumption was fairly low, as is shown in Table V. It will be noted that the percentage solids and the pumping rate were both low during this period, thereby resulting in a lower than normal solids transportation rate and correspondingly low-head losses, which permitted operation under these conditions. The velocity of 8.2 fps was only slightly above the safe operating velocity, and it is believed that a significant increase in the solids content of the pulp while using only one pump would have resulted in a lower velocity and stoppage of the pipeline.

The pumping efficiency $\left(\frac{\text{theoretical, hp-hr}}{\text{actual, hp-hr}} \times 100\right)$ varied from 41 to 54 pct according to the load on the pumps, and was lowest when the solids concentration of the pulp was lowest. All of the data obtained during the test were taken with velocities above the critical point, that is, within the turbulent flow region, so that the principal variables affecting the friction losses in the pipeline were solids content, velocity, and particle size distribution of the solids. From the data collected, the values for the Reynolds number were calculated for the various tests; these values are given in Table V. The Fanning's friction factor f was calculated from the actual head losses obtained from gage readings taken during the tests corrected for the change in elevation, see Fig. 6. The friction factors f were calculated from the equation,

$$H = 2f \frac{L}{D} \frac{V^2}{g}$$

Where: H = head loss in feet of fluid flowing (pulp); L = feet of pipe, D = pipe diameter in feet, V = pulp velocity in feet per second, and g = acceleration due to gravity in feet per second.

The Reynolds numbers are equal to

$$\frac{DVP}{w}$$

Where: D = pipe diameter in feet, V = pulp velocity in feet per second, P = pulp density in pounds per cubic foot, and w = viscosity of water at 70°F in pounds per second-ft ($w = 6.6 \times 10^{-4}$ lb per second-ft). The diameter of the pipe was 2/3 ft, and the length of the pipe was 16,000 ft. All of the calculations were based on the flow from the No. 2

Table V. Observed and Calculated Data From Test Operation of Hydraulic Transportation System for Tennessee Brown Phosphates

	Pulp Data					No. 1 Pump Press.			Head at Akin					
Period	Flow Rate, Gpm	Solids Content, Wt. Pct	Sp Gr[a]	Solids Rate, Tph	Pulp Rate, Tph	Inlet (Gage), Psi	Disch. (Gage), Psi	Devel. Head, Psi	Disch. Press. No. 2 Pump, Psi	Equiv. Ft Pulp	Total Head, Ft of Pulp	Pulp Velocity, Fps	Head Loss for Pulp, ΔHp, Ft/100 Ft	Fanning Friction Factor,[b] f
1	1344	12.7	1.082	47	368	4	103	99	148	316	420	8.6	2.62	0.0038
2	1372	16.3	1.126	63	384	4	106	102	172	389	476	8.8	2.98	0.0041
3	1421	11.1	1.080	42	384	4	103	99	142	304	411	9.1	2.57	0.0033
4	1383	14.0	1.102	53	382	4	126	122	148	310	417	8.8	2.60	0.0036
5	1423	13.3	1.096	52	390	4	108	104	152	321	428	9.1	2.68	0.0035
6	1357	16.8	1.125	64	382	4	112	108	167	344	481	8.6	2.82	0.0041
7	1317	11.9	1.084	43	356	4	105	101	139	296	403	8.4	2.52	0.0038
8	1288	10.7	1.075	37	348	4			109	234	341	8.2	2.13	0.0034
9	1344	16.6	1.123	62	377	4	102	98	164	337	444	8.6	2.77	0.0040
10	1323	17.2	1.128	64	372	4	127	123	161	330	437	8.4	2.73	0.0042
11	1481	17.4	1.129	73	418	4	128	124	181	371	478	9.4	2.99	0.0036
12	1359	14.4	1.105	54	376	4	124	120	162	339	446	8.7	2.79	0.0040
13	1447	15.8	1.116	67	424	4	123	119	181	375	482	9.1	3.01	0.0039
14	1282	17.3	1.128	62	361	4	120	116	191	392	499	8.2	3.12	0.0050
15	1388	14.3	1.109	57	385	4	117	113	189	395	502	8.8	3.14	0.0043
16	1343	0[d]	1.000	0					20[e]	46	150	8.6	1.41	0.0021
17	1460	0[d]	1.000	0					54[e]	125	229	9.3	2.16	0.0027

Table V. (continued)

Reynolds Number DVP/w ×10⁻⁴	No. 1 Pump		No. 2 Pump		Total Energy Input		Pumping Efficiency, Pct	Head Loss for Water, Ft/100 Ft, Hw	Hp[c]/Hw	Particle Size Distribution, Wt Pct of Solids			
	Oper. Time, Hr	Energy Input, Hp-Hr	Oper. Time, Hr	Energy Input, Hp-Hr	Hp-Hr Ton Solids	Kw-Hr Ton Solids				+6 Mesh	−6 Mesh +35 Mesh	−35 Mesh +200 Mesh	−200 Mesh
594	8.00	1672	4.30	885	5.5	4.1	45	1.96	1.34	5.1	6.8	57.6	30.5
632	8.00	1672	7.92	731	4.8	3.6	48	2.03	1.47	5.1	6.8	57.6	30.5
620	7.37	1720	4.49	349	6.2	4.6	46	2.17	1.18	3.3	10.8	54.0	31.9
612	8.00	1632	3.22	269	4.5	3.4	50	2.06	1.36	3.5	8.6	48.8	39.1
628	8.00	1592	4.32	350	4.7	3.5	52	2.18	1.23	4.9	12.9	50.8	31.4
611	8.00	1592	5.17	410	3.9	2.9	53	1.99	1.42	4.9	12.9	50.8	31.4
575	8.00	1672	4.77	387	6.1	4.6	41	1.89	1.33	9.2	9.2	48.4	33.2
556	0	0	8.00	1353	4.6	3.4	48	1.80	1.18	7.4	13.6	54.9	24.1
610	8.00	1672	4.40	478	4.3	3.2	48	1.96	1.41	8.5	12.0	54.6	24.9
598	8.00	1672	5.62	457	4.2	3.1	47	1.90	1.44	8.5	12.0	54.6	24.9
670	8.00	1672	7.25	646	4.0	3.0	54	2.34	1.28	4.3	12.6	57.2	25.9
606	8.00	1632	4.80	479	4.9	3.7	49	2.00	1.40	8.0	7.7	60.6	23.7
640	8.00	1672	7.79	816	3.6	3.4	53	2.24	1.34	10.2	10.2	52.0	72.4
582	8.00	1632	8.00	794	4.5	3.4	47	1.79	1.74	7.4	13.6	54.9	24.1
615	8.00	1632	7.01	781	5.3	4.0	51	2.08	1.31	8.0	7.7	60.6	23.7
541													
585													

[a] Based on true density of solids of 2.85.
[b] Calculated using Williams and Hazen formula with C = 100.
[c] Ratio of head loss with pulp to calculate head loss with water.
[d] Testing with water only.
[e] Data taken from No. 3 pump to Godwin.

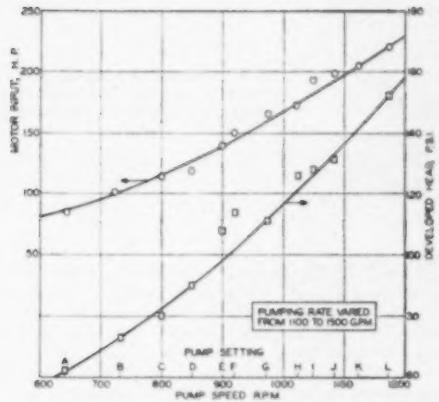

Fig. 7—Characteristic curves for No. 2 concentrate pump.

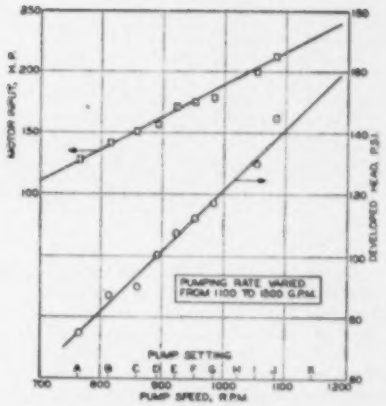

Fig. 8—Characteristic curves for No. 3 concentrate pump.

pump discharge to the pipeline discharge at Godwin. It may be seen in Fig. 10 that the friction factors obtained for conditions while pumping water were only slightly lower than those for the pulp normally handled in this system. Additional data are given in Fig. 10 to show the relation between the Reynolds number and the Fanning friction factor for water, as presented by other investigators.[9] It may be seen that most of the test data agree closely with the published data presented for steel pipe. The variations in the factors calculated from test data were attributed to variations in the concentration and particle size distribution of the solids in the pulp. Since pulps should show higher friction factors than water, it is presumed that the concentrate line was smoother, as a result of the polishing action of the solids, than the clean steel pipe used in the investigation reported by Caldwell and Babbitt. Calculation of the smoothness factor in the Hazen-Williams formula showed that when pumping water the factor C was equal to approximately 168. This value is considerably higher than the value of 140 reported as a normal smoothness factor for smooth steel pipe.[1]

Using a smoothness factor of 168, the head loss was calculated for conditions in which only water was pumped; the same flow rates were used as those observed during each of the test conditions in which concentrate pulp was pumped. This head loss for water at each rate, and its ratio to the corresponding head loss for pulp, $\frac{\Delta Hp}{\Delta Hw} = Y$, is shown in Table V. The ratios are also plotted in Fig. 11 with a curve representing data obtained in the experimental work with the 2 and 4-in. diam pipeline. Variations due to changes in velocity within the range studied and the particle size distribution of the solids are not correlated in this presentation, and these variations account for the scattering of the datum points. For example; the wide difference in the ratios shown for pulp containing solids within the range of 17.2 to 17.4 pct is attributed to the variation in velocity during three periods (periods Nos. 11, 10, and 14) which had velocities of 9.4, 8.4, and 8.2 and ratios of $\frac{\Delta Hp}{\Delta Hw}$ of 1.28, 1.44, and 1.74, respectively.

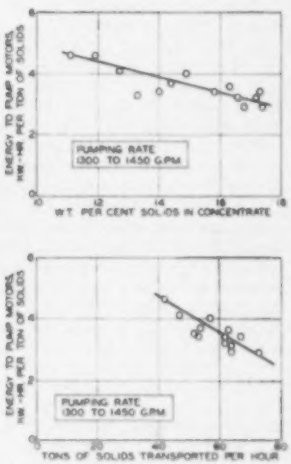
ENERGY TO PUMP MOTORS, KW.-HR. PER TON OF SOLIDS
PUMPING RATE 1300 TO 1450 G.P.M.
WT. PER CENT SOLIDS IN CONCENTRATE
TONS OF SOLIDS TRANSPORTED PER HOUR

Fig. 9—Energy consumed in pumping phosphate concentrate from Akin to Godwin.

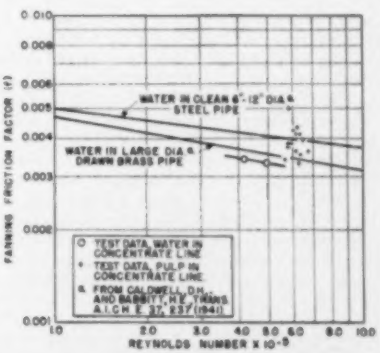
FANNING FRICTION FACTOR (f)
WATER IN CLEAN 6"-12" DIA. STEEL PIPE
WATER IN LARGE DIA. DRAWN BRASS PIPE
TEST DATA, WATER IN CONCENTRATE LINE
TEST DATA, PULP IN CONCENTRATE LINE
FROM CALDWELL, D.H. AND BABBITT, H.E. TRANS. A.I.Ch.E. 37, 237 (1941)
REYNOLDS NUMBER X 10⁻³

Fig. 10—Fanning friction factors vs. Reynolds numbers for concentrate pipeline.

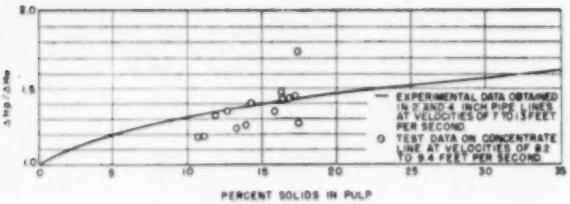

Fig. 11—Effect of solids content of pulps on ratio of head loss in pipelines when pumping pulp to head loss when pumping water.

The experience of the Tennessee Valley Authority with the hydraulic transportation of phosphatic concentrates has demonstrated the feasibility of using this method to transport phosphates under the conditions existing in its phosphate beneficiation plant. Transportation costs have compared favorably with the costs of more conventional methods, and operation of the system is not limited by factors, such as bad weather and poor roads, which characteristically interfere with other methods, such as trucking. Although this system for hydraulic transportation lacks some of the flexibility of other transportation methods in that its operation must be integrated with the concentrating and classifying operations, this has not been a disadvantage in the TVA operations. In other words, the pipeline is not operated unless the processing units at Akin and Godwin are operated, and similarly, the processing units cannot be operated unless the pipeline is operable. This, of course, prevents the accumulation and storage of surplus material at points beyond the washer to provide for emergency operation of subsequent units, a disadvantage not encountered with other methods of transportation. Thus, it is apparent that with any hydraulic transportation system, the process steps immediately preceding and following the pipeline are important factors in its design.

Operating data on the TVA pipeline show that it is not normally operated at its maximum practical capacity. Although the solids content of the pulp is usually in the range of 12 to 17 pct, concentrations of 20 to 25 pct may be handled readily. Reaching the higher concentration in the TVA system while keeping the flow rate in the pipeline above the critical velocity would entail operating the related process steps at higher rates; this has not been considered expedient. Of course, a pipeline of smaller diameter, and consequently lower cost, could be used satisfactorily with pulp solids concentrations in the upper range. However, since a pulp containing 12 to 17 pct solids also contains about the required amount of water for the operation of the classifying equipment at Godwin, and since the problem of obtaining an adequate water supply is greater at Godwin than at Akin, it has been found desirable to pump the necessary water for proper operation of the classifiers from Akin along with the phosphate concentrate.

After replacing the original wood-stave piping with abrasion-resistant steel pipe, maintenance of the pipeline became a minor problem. Since wear of the pipe walls is greatest on the bottom of the pipe, the sections of pipe are turned periodically to equalize the wear on all sides. This results in maximum life from the pipe and reduces replacements to a minimum. Turning of the pipeline is a regular maintenance job and is carried out, a few sections at a time, on a prearranged schedule coordinated with scheduled shutdowns of other parts of the plant. Wear has also been noted to be greater at curves or bends, as might be expected. When leaks occur in the pipe, they are repaired in the field by means of clamps which encircle the pipe, holding a rubber gasket over the leak. Wear on the pump housings is not serious but replacement of pump impellers is a regular maintenance item. Wear and replacement of the pump bearings is also a significant factor. The frequency of replacement depends upon the adequacy of the seal-water supply and upon keeping the seal-water pressure above the internal pump pressure to prevent passage of sand into the bearings. Although this was a major source of difficulty in early operations because of an inadequate seal-water supply, improvement of the seal-water supply system to correct this deficiency has greatly alleviated the bearing problem.

In normal operation of the pipeline, two pump operators are required, and the nominal pumping rate is 40 to 50 dry tons of concentrate per hr. During 1949 and 1950, when pumping concentrate was at the rate of about 350,000 dry tons annually, the cost of operating the hydraulic transportation system was about $0.24 per dry ton of material handled. Approximately 50 pct of the cost was for direct labor, 8 pct was for power, 34 pct was for maintenance, and 8 pct was for depreciation. This amounts to about $0.08 per dry ton mile for transporting the approximately 3-mile distance between the Akin and Godwin plants. However, in comparing this cost with costs for other methods of transportation, it should be kept in mind that the TVA system eliminates certain handling operations that would exist at both locations if other methods of transportation were used. Thus, it is only fair to credit the above transportation cost with the savings involved in the elimination of these handling costs.

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Contact Angles and Surface Coverage

by W. Philippoff, S. R. B. Cooke, and Donald E. Cadwell

THE importance of contact angles in flotation has long been recognized, but little has been done to get quantitative relationships between the surface coverage of the mineral by the reagent, the length of the hydrocarbon chain, and the contact angle. It is well known that for a complete monolayer on the surface of a mineral the contact angle θ is determined only by the reagent[1] and principally by the chain length of the hydrocarbon radical. Although for flotation the monolayer usually is regarded as being complete, early experiments of Wark and Cox[2] showed that at very small concentrations of potassium ethyl xanthate the contact angle on galena changed in a continuous way from 0° to the equilibrium value of 60°. Further, Gaudin and Vincent[3] showed a similar continuity using heptylic acid and sphalerite. This shows that with probably incomplete monolayers, θ varies with the degree of surface coverage. At the time these experiments were made it was impossible to measure the actual amount of surface coverage, especially since the decrease of the collector concentration in the liquor could be influenced by chemical side reactions. Through the use of the radioactive tracer method for the measurement of the amount of collector adsorbed on the surface of the mineral, Gaudin and Bloecher[4] were able to determine accurately the actual surface coverage of dodecylamine acetate on quartz, but they did not measure any contact angles. It is believed that the relationship between the surface coverage and the contact angle is extremely important in flotation, especially as Gaudin and Bloecher showed that a 95 pct recovery is possible with a coverage of only 5 pct.

Theoretical Considerations

The work of adhesion W is the energy bound (in ergs per sq cm) when an interface is formed. It is determined by the difference of the free surface energies (surface tensions) of the respective phases γ_1 and γ_2 before contact and the interfacial free surface energy γ_{12} after contact. The definition of the contact angle permits the elimination of the unknown surface energy of the solid and hence the calculation[5] of W from the surface tension of the liquid γ and the contact angle θ:

$$W = \gamma\,(1 + \cos\theta) \qquad [1]$$

Consider a surface consisting of two components, identified as 2 and 3, 1 being the liquid. The amount of 2 in the composite surface is x, where $0 < x < 1$. The respective contact angles with 1 and air are θ_2 and θ_3. The total work of adhesion of such a surface is the sum of the values of the components multiplied by their respective areas. A formal relation can be written:

$$W_m = x \cdot W_2 + (1 - x) \cdot W_3 \qquad [2]$$

in which W_m is the work of adhesion of the composite surface. Eq 1 can be introduced into W_2 and W_3, but in the case of W_m some discussion is needed. If the two components are segregated, each component forming an integral part of the composite surface there is *no* physical meaning for a contact angle θ_m calculated from W_m by using eq 1. If, on the other hand, the area of each *patch* of the components is small in comparison with, say, the diameter of the bubble used in measuring, it could be expected that θ_m would have a physical significance, because of the action of the surface tension of the liquid-gas interface tending to smooth out sharp corners. How small these patches would have to be is difficult to predict, but by using composite surfaces with microscopically

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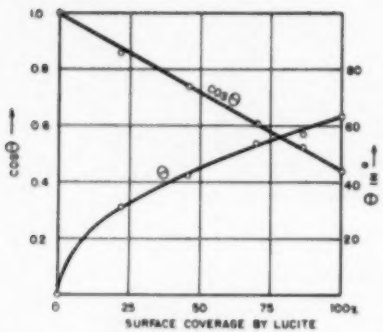

Fig. 1—Results of experiments on the composite surfaces of glass and lucite. Each value on the curve is a mean of five measurements at different places on the composite surface.

measurable patches a direct experimental approach is feasible.

Using this restriction, eq 1 can be introduced into eq 2 and, after rearrangement, gives:

$$\cos\theta_s - \cos\theta_m = x\,(\cos\theta_s - \cos\theta_t) \qquad [3]$$

For the practically important case of a hydrophobic film of material 2, having a contact angle θ_t at 100 pct coverage and a hydrophilic substrate 3, for which $\cos\theta_s = +1$, the very general eq 3 can be simplified as follows:

$$1 - \cos\theta_m = x\,(1 - \cos\theta_t) \qquad [4]$$

The meaning of this equation is that if $\cos\theta_m$ is plotted against x, the graph is a straight line descending, with increasing values of x, from 1 at $x = 0$, the limit being $\cos\theta_t$ at 100 pct coverage.

It is now possible to visualize what happens to the hydrophilic surface when the equilibrium concentration of the collector is increased gradually. At first it is natural that only very few collector molecules are adsorbed. This is equivalent to a gaseous film at the interface in which the molecules are independent of each other, and the gas law $p \cdot v = RT$ is valid. In the case of surface films: (surface pressure in dynes per cm) × (area per molecule in sq A) $= kT \cong 400$ at room temperature. All the evidence as summarized by Adam[4] indicates that even with the longest hydrocarbon chains used (22 carbon atoms in the chain) the molecules lie flat on the surface of water.

The complete monolayer is considered to be formed by molecules standing up, their cross-section independent of the chain length being 20.5 sq A. Logically, it might be expected that there would be a transition between the gaseous and the closely packed film, in which the originally horizontal molecules assume a perpendicular position.

On the basis of eq 4 it is possible to calculate the conditions to be expected if a hydrophilic surface is covered with hydrocarbon molecules, the cross-sectional area of each molecule being F_m. A value of from 104° to 109° for the contact angle of paraffin (θ_t) can be assumed. The amount of collector necessary to form a complete monolayer is given by the following calculation:[4]

The area of 1 molecule is F_m sq A
The area of 1 mol is $F_m \cdot N$, where N is the Avogadro constant
The area of 1 g of collector is $F_m \cdot N/M$, where M is the molecular weight of the collector
The area of 1 g of mineral is S, the specific surface, here in sq A per g
Then for complete coverage:
Grams of collector/grams of mineral $= S \cdot M/N \cdot F_m$

The surface coverage is given by the ratio of the amount actually adsorbed to the calculated value for the complete monolayer: this means that eq 4 can be written as:

$$1 - \cos\theta_m = N \cdot F_m/S \cdot M\,(1 - \cos\theta_t) \cdot \frac{\text{grams of collector}}{\text{grams of mineral}} \qquad [5]$$

Using the values of Gaudin and Bloecher, eq 5 can be evaluated. $F_m = 23.4$ sq A, the area of an active site of quartz, as calculated by Gaudin; $M = 245$, the molecular weight of the dodecylamine acetate; $S = 2340 \times 10^{16}$ sq A per g, the specific surface of the quartz powder; $\theta_t = 106°$, the contact angle of paraffin. This gives in convenient units:

$$1 - \cos\theta_m = 1.28 \cdot \frac{\text{micrograms of amine/gram quartz}}{401}$$

$$1 - \cos\theta_m = 0.00319\,(\text{micrograms of amine/gram quartz}) \qquad [6]$$

$$1 - \cos\theta_m = 1.28 \cdot x \qquad [7]$$

Eq 6 would mean that a contact angle less than that of paraffin is determined only by the percent coverage of the surface by hydrocarbon (paraffin). As an example, an angle of 20° as calculated by the present authors, from measurements of Coghill and Anderson[7] on a highly cleaned and dried polished glass specimen, is determined by a coverage of only 5 pct. If the contamination were paraffin, this means that only 5 pct of the surface would be covered. This infinitesimal amount of matter, of the magnitude of 10^{-6} g per sq cm, would make the known sensitivity of dry hydrophilic surfaces to impurities easily understood. On the other hand, the contact angle on built-up layers of Ba-stearate, as investigated by Bikerman,[8] was 90°, corresponding to about 78 pct coverage by paraffin. For crystals of long-chain acids and alcohols, θ is 100°,[9] corresponding to a coverage of 92 pct by paraffin. These results mean that a complete monolayer of a collector on a hydrophilic surface does not ensure a hydrophobicity in the sense of a paraffin coating.

A distinction between paraffin coverage and collector coverage must therefore be made. It is known that θ depends, even at very high collector concentrations, on the length of the hydrocarbon part of the collector molecule.[5] For instance a monolayer of potassium ethyl xanthate gives an angle $\theta_t = 60°$, corresponding to $x = 39$ pct, meaning that this layer is only as active as a coverage by paraffin of that magnitude should be. In this case the polar part of the collector molecule causes the monolayer to behave as if 61 pct of the surface were hydrophilic. The longer the hydrocarbon chain, the more nearly the value for a paraffin coverage is attained. Short chain collectors apparently cannot inhibit completely the attraction of the water molecules to the mineral surface.

Using an appropriate experimental value for θ_t (for complete coverage) the relative coverage x and θ_m can each be calculated, knowing the other one, by using eq 4.

It has often been stated that the absorbed layer on a hydrophilic surface is in equilibrium with the collector in solution. The partition coefficient between the surface and the solution seems to depend on the pH. Kellogg and Vasquez-Rosas[11] have shown that at a constant collector concentration the contact angle of dodecylamine chloride on quartz can range from nearly 0° to about 80°. At pH 6.5 their value was 10°, corresponding to a coverage by paraffin of 1.3 pct; at a pH of 10, θ was 80°, or x = 64 pct. This change of the quantity of *adsorbed* collector at a constant collector concentration at different pH values, which follows from the theory evolved here to explain the change in θ at different pH, has not yet been investigated experimentally.

The adsorption of an amount of collector equivalent to a complete monolayer does not necessarily mean that a uniform coating is present, for it is equally possible that a patchy polymolecular adsorption occurs. The measurements of θ permit the calculation of the equivalent actual coverage by paraffin, independent of whether the adsorbed layer is mono or polymolecular, complete or incomplete.

Experimental Approach

To check eq 4, some experiments were made on composite surfaces of glass and lucite. Fine glass fibers were packed, similarly to the bristles of a paint brush, in a hole drilled in a piece of brass, then mounted in lucite by polymerizing the monomer around the fibers. Sections of different surface coverages x, determined by the ratio of the lucite to the total surface were made by cutting across the fibers at different distances from the brass. These sections were moulded in lucite in a press and polished on a lead lap with alumina on a Graton-Vanderwilt machine. After cleaning with carbon tetrachloride, then lightly buffing on a cloth lap with −3 micron quartz in distilled water, they were tested with an air bubble in the same bubble machine used for the quartz specimens.

The only case where the amount of coverage probably is known exactly is that of quartz and dodecylamine acetate, investigated by Gaudin and Bloecher. The writers relied on their determinations of the amounts of collector adsorbed and measured θ for different collector concentrations. A bubble machine was used, the contact angles being measured by means of a position-angle ocular. The precision of each experiment is estimated at ±0.5°. Samples of crystalline quartz, mounted in lucite, were polished on a cloth lap with distilled water and fine quartz to eliminate any possibility of contamination, and were tested under water for absence of a contact angle. They were conditioned for periods of some hours in about 200 cc of the amine solution, making sure that equilibrium of the contact angle was obtained. To attain this, tapping the holder and long observation times, sometimes more than 24 hr, were needed, thus making the investigation time consuming. The change in collector concentration due to adsorption on the approximately 1 sq cm of quartz surface was disregarded. For the range of amine acetate concentrations used in the present paper, 1 to 300 mg per liter, Gaudin and Bloecher's work covered the pH range of 5.83 to 6.33. In the present investigation the pH was maintained at 6.5 ± 0.2.

The following procedure was used to obtain the surface coverages x from the measurements of Gaudin and Bloecher. In the range through which their square root relationship is valid, up to 200 mg per liter, the proportionality constant between the square root of the concentration in mg per liter and the quantity of amine adsorbed in micrograms per gram of quartz (having a specific surface S = 2340 cm² per g) was calculated for each of their experimental points. The mean of these constants, 6.95, was used to calculate the adsorbed quantities at the concentrations taken for the contact angle measurements. The surface coverage x is the ratio of the calculated adsorbed quantity and 401 mg per g, the value for the complete monolayer, as given by eq 7.

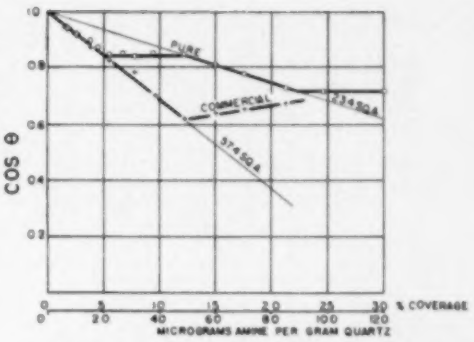

Fig. 2—Results on the quartz specimens. The curves for both the pure and commercial sample are given.

Results

The results of the experiments on the composite surfaces of glass and lucite are shown in Fig. 1. The angle θ_0 for lucite of that polish was determined experimentally as 63½°. Each value on the curve is a mean of five measurements at different places on the composite surface. No time dependence was noticeable.

The results on the quartz specimens are summarized in Fig. 2, the curves for both the pure and the commercial sample being given. In the range up to 5 pct coverage, calculated on the assumption that the molecules are perpendicular to the surface, both curves practically coincide. In the range from 5 to 12 pct coverage, corresponding, to an equilibrium concentration of from 8.3 to 50 mg/liter for the pure amine, θ remains constant at 32° within ±1°. Then θ increases (cos θ decreases) to about 22 pct coverage (150 mg/liter), to remain apparently constant with an angle of 44° to 45°. The commercial sample apparently shows the first transition beginning at 50 mg/liter. Its curve then merges with the curve for the pure sample, θ decreasing with increasing concentration. This has yet to be confirmed exactly. Experimental difficulties in more concentrated solutions arose from the lack of enough pure material and from the crystallization at concentrations above 200 mg/liter of some impurity in the solutions of the commercial sample. In any case the range of concentrations occurring in practice was fully covered.

Discussion

The experiments on the composite surfaces, where there is no doubt regarding the degree of surface coverage, have confirmed eq 4. Equally the linear relation between cos θ and the calculated surface

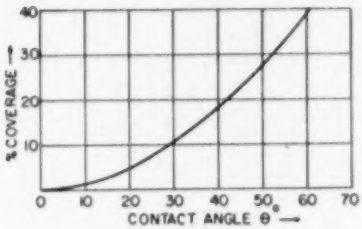

Fig. 3—Equation 6 used to compute surface coverages by paraffin from measured contact angles.

coverage x for the quartz experiments is valid through a large range, especially for the solutions of the commercial amine. In the case of quartz however, the angle for complete coverage by paraffin, 106°, was not attained.

These results form a basis to discuss contact angle phenomena using eq 7, connecting the surface coverage by paraffin to the contact angle of the composite surface.

The experimental slope of the graph between 50 and 150 mg/liter of the pure amine checks exactly with that calculated from eq 7. This means that in this region the molecules conform to the concept of being perpendicular to the surface. By evaluating F_a from the initial slope, 57.4 sq A is obtained instead of 23.4, the area of the active site. As the molecules lie flat in the gaseous film, the ratio of 57.4/20.5 = 2.80 represents the relation between the cross-sectional area and the axial section, 20.5 sq A, of the amine molecule. Taking an estimated breadth of the hydrocarbon chain of 4.4 A, a chain length of 13A is obtained. Eleven C-C bonds would give 14A, so that the value of 13A seems very probable. This check would seem to prove that this gaseous film, which is 2.45 (= 57.4/23.4) times as active on the surface as the layer of perpendicular molecules, is the one actually used in flotation with good results. Gaudin's recovery of 95 pct corresponds to the end of the gaseous film region.

The area where a constant θ is reached corresponds to 5 pct coverage, giving 472 sq A per molecule, equivalent to a surface pressure of only 0.85 dynes per cm, assuming the gas laws to be valid. It should be noted that Harkins et al[11] found such first order transitions in investigating the adsorption of heptane on iron oxide, silver, and graphite from the gaseous phase. The beginning of these transitions occurred at 400 to 600 sq A per molecule, comparable to our value of 472.

This check, together with the general discussion previously made, indicates that the molecules are adsorbed lying in the surface up to about 470 sq A per molecule, then begin to stand up perpendicularly to the surface. The surface coverage with hydrocarbon chains remains approximately constant until an area of 472/2.45 sq A = 193 per molecule is reached. The amount of adsorbed collector increases in this region, but the coverage by paraffin chains remains essentially constant. The behavior of the layer then follows closely the theory of molecules perpendicular to the surface to a point of saturation beyond which the coverage, according to this theory does not increase any more. In fact, only about 22 pct of all possible sites then are occupied by the dodecylamine molecules.

The accordance permits the assumption that eq 6, which gives a quantitative relationship between any contact angle and the surface coverage x by paraffin, has a physical meaning and can be used to compute surface coverages by paraffin from measured contact angles, as shown in Fig. 3. This also permits the possibility of evaluating the efficiency of collectors when the total amount added and the amount active on the surface from the measurements of θ are known.

There are further aspects of the activity of compounds of different chain lengths. In the range of the validity of the theory for the perpendicular molecules, at concentrations of 50 to 150 mg/liter, the curve is probably independent of the chain length. The curve or its continuation should form a limit at some as yet unknown concentrations for amines of different chain lengths. Longer chain amines, such as hexadecylamine, have a greater ratio of length to width than dodecylamine. For the same surface coverage, as calculated on the assumption of perpendicular molecules, the values of θ should be greater than those for the dodecylamine at low concentrations of the collector. Logically, the shorter amines (octyl and decyl) should not show as much of a difference between the slopes of the two lines as the dodecylamine. This is easily tested experimentally.

Practically, this investigation has proved that *any* contact angle between 0° and 45° can be realized in a predetermined way for the system dodecylamine acetate-quartz. This opens the way for the experimental determination of θ necessary for the flotation of certain particle sizes.

From the results of Gaudin and Bloecher, it is calculated that a recovery of 95 pct for their particular size corresponding to S = 4340 sq cm per g, was obtained at an actual surface coverage by paraffin of 12 pct and an angle of 32°. A recovery of 10 pct was achieved at 1.3 pct coverage with an angle of 10°. This gives an approximate range of the contact angles necessary to float quartz of that fineness.

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Development of The Disco Process Of Low Temperature Carbonization

by C. E. Lesher

THE Disco* process for the production of low temperature coke and its immediate predecessor, the Wisner or Carbocite process have been described in the voluminous literature of low temperature carbonization.[1-4]

This paper will describe the development of the process from the first attempt at commercial operation by the American Gas and Electric Co. at Philo, Ohio,[5] in the 1920's to the successful operation in 1949 by the Disco Co., a division of the Pittsburgh Consolidation Coal Co., of the 1000-ton per day Disco plant near Pittsburgh, see Figs. 1-2.

In the 1924 coal industry depression following the hectic post World War I expansion, the northern Appalachian, unionized bituminous coal fields were affected first and foremost, and Western Pennsylvania as hard as any. Competitive market pressure was heaviest on the lower grades of steam coals, notably those from the Panhandle district where ash and sulphur are inherently higher than in the same Pittsburgh coal bed to the east along the rivers. The firm structure of Panhandle coal while favoring the prepared sizes, depressed the usefulness and value of the finer sizes. Panhandle slack and nut slack were marketed with difficulty at consistent, and at times considerable, loss.

Many solutions were suggested, and the Pittsburgh Coal Co. investigated all. Lower delivered prices through lower mine costs and more advantageous freight rates were obvious goals, but equally obvious were the difficulties of self attainment of either in any substantial measure. Selling Btu's in pipes was abandoned as an idea after a thorough study of the economics of distribution of what was then, and is even now, the only developed continuous process of complete gasification of bituminous coal-producer gas.

Although the coal company purchased huge blocks of electric power, it sold not a car of coal to the central power stations. Public utilities controlled the markets for electric current, and it was found that no matter at how low a cost current might be put on the bus bars by the coal company, distribution to its widely scattered coal mines made such an enterprise practically impossible.

Consolidation of underground operations and a huge modernization program of mining operations in the 1920's were but preparation for the introduction of mechanical cleaning of the coal for market, particularly the smaller sizes of steam, gas, and by-

* Disco is the registered trade name of a solid fuel made by the Disco process of low temperature carbonization of coal.

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Fig. 1 (top)—Plus 2-in. Disco from the new plant, with the largest pieces about 8 in. This view was taken at the base of a 10,000-ton stockpile.

Fig. 2 (bottom)—The 1200-ton per day Disco plant under construction. The rectangular structures above each retort are roaster-storage units in which the coal and recycle is preheated before charging into the carbonizers.

Fig. 3—Thermodizer, at left, and carbonizer, each 3 ft diam by 50 ft long, in the Wisner pilot plant under construction in 1932.

product coals. Mechanical cleaning was a necessary step in anticipation of mechanical loading at the face. Upgraded Panhandle slack, however, continued to be a difficult product to sell. In the meantime mechanical stokers, large and small, were coming to the fore and by splitting the 1¼ in. slack at about ⅜ in., a double screened stoker size was made available on a large scale and at substantially better market realization than for slack. But this left the − ⅜ in. size to compete with more easily grindable coals with lower ash and sulphur from other fields. The problem was narrowed down to the economic disposal of this fine Panhandle slack. If the − ⅜ in. coal could be sold in some form and return no more than total, overall cost of mining and cleaning, the cleaned, carefully screened, and prepared larger size of this firm structure coal would carry the load and there would be an overall profit for the enterprise. This was, and still is, the economic incentive for processing this coal.

Low Temperature Carbonization

The Pittsburgh Coal Co. produced and sold solid fuel. Carbonization was the obvious route to explore for converting this fine coal to larger sizes suitable for hand firing. Full oven tests of several sizes of steam slack were made in Koppers ovens at Chicago and at Philadelphia. The coke was fingery and small sized. Yields were not encouraging; the initial investment was high; and selling the surplus gas and other byproducts was alien to the coal company. Furthermore, high temperature coke, although an accepted smokeless hand-fired domestic fuel in other sections of the country, would, it was believed, meet serious sales resistance in the markets where the coal company sold its lump and preferred sizes of high volatile bituminous coal because of differences in ignition and burning characteristics.

It was not difficult to write the specifications for the desired fuel that was to be made from this smaller size. It must be dense, firm, clean, smokeless, easily ignitable, and able to stand stockpiling. Above all, the cost of making it must be such that it could be sold competitively with other smokeless solid fuels. With compromises here and there along the line, those objectives have been met with Disco.

Low temperature carbonization of coal had just about passed the zenith of public acclaim and common stock financing when in 1928 the Pittsburgh Coal Co. first seriously considered it. There was little to encourage a venture into the field, for the record was a succession of failures. However, there seemed to be a firm foundation for a start, and with the backing of its Board of Directors and support of A. W. Mellon, management of the Pittsburgh Coal Co. authorized a beginning in 1929.

Two early decisions narrowed the field and simplified the economics: no process was to be given serious consideration if it gave a solid product requiring briquetting, and the sole byproduct to be valued would be tar, and that only at its fuel value. Three processes were selected for study: the Plassmann in Germany, the Illingworth in Wales, and the Wisner in the United States. For each there was already a sizable pilot plant in operation and at each, in 1929 and 1930, substantial quantities of coal were processed and the products given consumer trials. Not one of the three pilot plant tests was encouraging, but the Illingworth, a vertical cast-iron retort, made a good coke. Rights for this process were secured, detailed engineering estimates were made for a large installation, and in 1931 a pilot plant with about 3 tons per day capacity was built and operated for nearly a year. Low capacity, high cost and intricate construction finally ruled out this as well as the Plassmann process.

The Wisner *coal ball* process was adopted for development. This paper is concerned primarily with the problems encountered in building the first commercial, low temperature coking plant in this country and so far as known, the only continuous, economic operation anywhere. Starting with a pilot plant in 1933 which grew to a three-unit semicommercial operation in 1936, and in 1940 processed 100,000 tons of highly coking coal, the project then stalled until, under the farsighted and able management of the Pittsburgh Consolidation Coal Co., a full-scale plant was authorized in 1947, see Fig. 2. This plant, costing over 3 million dollars, was in full operation by the end of 1949, and has been since then processing its design quota of over 1000 tons of coking coal a day and to this extent relieving the market of a fine coal product from the Champion Preparation Plant of the coal company and upgrading it to a premium, smokeless household fuel. There has never been a time since the first plant was started that 1000 tons of coal to the Disco plant did not give the mines an extra 4000 tons of operating time.

The Disco plant is a part of an integrated coal company operation covering mining, preparation, and sale of solid fuel. Of the three elements in the economics of this operation, conversion cost alone is the concern of the coke plant management, for the price charged in the books of account for coal and the realization for products are determined by markets and overall company policy. Of conversion cost, that which is direct is controllable from day to day—labor, supplies, power, and maintenance. Property and payroll taxes, insurance, reasonable amortization, and share of top company overhead are more or less fixed. From the receipt of 4 pct moisture coal to the loading of Disco into railroad cars and dehydrated tar into tank trucks, the cost of hourly labor is 1 man-hour for 2.5 tons of coal processed. This includes maintenance labor but not supervision. For 1 ton of coal processed, there is purchased approximately 20 kw-hr of power, a relatively small quantity of water and of natural gas for pilot lights.

Operating and maintenance supply costs are between 5 and 6 pct of plant investment.

Wisner Pilot Plant

A Wisner pilot plant for from 10 to 12 tons of coal per day was built and operated on the same site and simultaneously with the Illingworth, see Fig. 3. C. B. Wisner, inventor and Walter Keenan, chief engineer of Foster Wheeler Corp., who was representing the owners of Wisner's patents at that time, designed the retorts and flowsheet according to Wisner's specifications, which followed the design of the Philo plant. Until his death in November 1932, Mr. Wisner assisted in starting the pilot plant. Two 3x50-ft retorts, each with an annular heating space between inner and outer rotating steel shells, were thermodizer and carbonizer respectively. Heating was indirect through the inner steel shell from hot flue gases traveling in a closed circuit through the annular spaces, circulating fans and gas fired heating furnace. In all essentials this process heating system has been used in subsequent installations.

Feed coal entered the high end of the thermodizer, where it was dried and conditioned for carbonizing by heating in air to 600°F. From the low end of the thermodizer, the coal was carried in a screw conveyor to the high end of the carbonizer, where it was distilled at 800°F and was discharged as coal balls. Tar was collected and stored; gas was wasted. The unit was heated with natural gas. The charge material was − 10 mesh coal, containing oil flotation concentrates from the Champion Preparation Plant of the company.

The experience gained in processing 500 tons of $\frac{3}{8}$ x0 coal at Philo, Ohio, in 1929 provided a starting point for the work in the new pilot plant. At Philo there had been short periods during which coal balls were produced, but there were longer periods when the retorts were plugged with masses of coke or the machinery failed. As work progressed at the pilot plant, it became obvious that the trouble was lack of control at every step, but more important, too little was known of what should be controlled.

Certain problems were considered fundamental. Feed coal to the rotating preheater would not flow down the retort according to schedule. It packed and built up in the shell; it did not discharge regularly either as to quantity, temperature, or moisture content. A surge bin for intermediate storage between the thermodizer and carbonizer to equalize the feed to the latter was ineffective because the hot coal ignited, coked, and would not flow or feed.

Once in the carbonizer the preheated and preoxidized coal became unpredictable. At times the coked product was in small balls or just fines, again in masses too great to pass the exit throat of the carbonizer, see Figs. 4 and 5. That is, this pilot plant carbonizing coal in a revolving retort was behaving like all others before it.

In England several attempts had been made to make low temperature coke in revolving kilns and in every case final resort had been to devices in the drums to break up the scale and crust that always formed. None were known to have succeeded. The K.S.G. retorts when brought to this country from Germany by the International Combustion Corp., failed because the plastic coal plugged the retort. The first few months' operation of the Wisner pilot plant at Champion confirmed the experience of others with rotary retorts, but it was found that, as Wisner claimed, the proper, and just the proper,

Fig. 4—Strange shaped coke made in the first retort before it was learned how to control the operation.

amount of preliminary oxidation would prevent excessive encrustation on the retort walls.

In fact, it was found that the precision in roasting required to maintain steady carbonizer operation and production was beyond practical limits. Laboratory tests were developed to show and permit control of the amount of oxidation of the coal prior to feeding it to the carbonizer, but it was found that it required 2 hr in the laboratory for a determination that the carbonizer itself would give in 50 pct that time. It was apparent that some immediate and effective control of the feed to the carbonizer was essential if there was to be smooth operation and a product of good quality, and that preoxidation alone was not a dependable way to accomplish that end.

It will be recalled that Wisner had placed reliance on preoxidation and declared[6] that "coal blending is not a satisfactory answer" to the "rolling up of softened coal into balls too big to get out of the gastight container"—the rotary retort. On the contrary, Roberts,[7] Wheeler, and others advocated blending of noncoking coal, anthracite, and coke fines with coking coal to control the plastic property of the mixture in rotary retorts. Reduction to practice of the theory of blending however, appears to have been confined to the Davidson retort in Great Britain, where, according to Roberts, by proper temperature

Fig. 5—Another sample of large irregular masses made in the first retort.

Fig. 6—The first commercial Disco plant. Built in 1933 on the site of the pilot plants, the carbonizer was 6 ft in diam and 90 ft long, and produced 60 tons of Disco per day for 15 years.

control it was possible with the aid of scrapers to keep the shell quite free from carbon crusts.

Wisner Process Departures

As the first departure from the Wisner process, experiments were made of blending. The theory on which extensive tests were made at the pilot plant in the winter of 1932-1933 was that the ball size of product could be controlled by the addition of breeze to reduce, or of fresh coal to increase, the size. Auxiliary bins with individually controlled feeders were installed, one containing pulverized low temperature coke, the other fresh coal. Neither was preheated. Small amounts of one or the other were added to the main stream of preheated and partially oxidized coal as it entered the 3 ft carbonizer. The results were highly satisfactory. For the first time the operators were able to control the product of the carbonizing retort. This method has in principal become a part of the Disco process, except that the control has been confined to varying the percentage of recycle breeze, with this breeze preheated.

The second departure from the Wisner flowsheet was the introduction into the circuit between the thermodizer and carbonizer of a storage or surge bin. Early experience demonstrated the necessity of a means of regulating the irregular flow from the preheating retort to a steady flow into the carbonizer. The simple solution of a surge bin was a failure because the hot, fine coal either caked or took fire. The solution was found in a loaded conveyor, which was in effect a surge bin in which the hot coal was kept in motion and from which the desired quantity was withdrawn under controlled conditions and fed into the carbonizer. In this storage the hot carbonizer feed was mixed, heated uniformly throughout, and from it a reliable feed, both as to rate and quality, was possible.[9]

The third important departure from the original design was the development of the rabble hearth for preheating and oxidation of the coal. The rotary kiln for drying and preheating was unsatisfactory. Heat transfer was limited; the coal packed and caked on the walls; the temperature and moisture content was exceedingly irregular. A small, four-deck, rabbled preheater built and operated as the last important step in the operation of the pilot plant solved this problem. In May 1933 the pilot plant was torn down to make way for the installation of the first commercial sized retort.

The pilot plant development had demonstrated that with adequate controls, the rotating retort would make a satisfactory low temperature coke and that adequate controls were possible with the rabbled hearth, the intermediate storage, and recycle of breeze.

Commercial Plants

Three commercial sized units were built in the period 1933 to 1936 and operated continuously until dismantled in 1949 when the present plant went into production. These units have been described in detail[9-12] and reference will be made here only to the steps in the development from pilot plant to this first commercial operation and then to the design of the present plant.

The first unit, started in November 1933, see Fig. 6, had a four-deck, rabble-hearth roaster and a two-deck storage in which buckets carried coal around the circuit, and from which a constant amount was withdrawn for carbonizer feed, and of which a part was recirculated to the roaster. This piece of equipment had a remarkable record. Except for a few wrecks caused by drive shaft failures, this roaster-storage combination was almost trouble free. The original heavy rivetless chain (with pin renewals, of course) lasted the life of the unit; high carbon steel track installed in 1934, original buckets and rabbles, the casing, top, bottom and sides, hearth plates, and heating gas ducts, were all intact and in perfect condition when dismantled in 1949.

However, results with the No. 1 carbonizer were different. It was designed, as were the Philo unit and the 3-ft pilot plant, with integral concentric revolving shells, insulated outside with slag wool mattresses. No way was ever found to hold this insulation for more than a year. The tires and drive sprocket were positioned in from the ends, which necessitated hot gas hoods each with two heating gas seals. Maintenance was high, but the unit continued to produce good Disco to the end.

A radical change was made in the design of the second unit. Tires and drive were mounted on the ends of the retort, which eliminated two heating gas seals. The outer shell was stationary, the inner alone rotating. Both original cost and maintenance were reduced materially. For the second and third units the roasters were the Wedge-type, all steel, circular, with nine decks, each deck above a heating gas duct. The rabbles were on radial arms attached to a vertical shaft. Although these machines gave satisfactory service, their high initial cost ruled them out for future installations. The same is true of the revolving double shell retort designed for the storage conveyors for these units.

There were many things learned in the operation of the first plant in which, in the years from 1939 to 1948, few alterations in plant or process were made. By the end of 1939 those who managed and operated the Disco plant were satisfied that the development had reached the stage where expansion was desirable, either by addition of more units or by a new installation. During World War II such construction did not appear to be feasible. The economic incentive remained however, and after the war the Pittsburgh Consolidation Coal Co. authorized construction of the seven-unit plant. This was completed in 1949, and the old plant was dismantled and sold for scrap in 1950.

Starting with one unit and production of 816 tons of Disco in 1933, this plant processed a total of 1,155,400 tons of coal in the years 1934 to 1948; produced and sold 829,400 tons of Disco and 15,834,000 gal of tar. The last month it was in full operation, December 1948, this plant had its record output of 8172 tons of Disco.

Open Ends, Dams, and Lifters

Important as were the control methods developed at the pilot plant, two discoveries made in the early operation of the three-unit plant were of great importance; both contributing to the final success of the process. One discovery was that by leaving the discharge end of the carbonizing retorts open to the atmosphere and controlling the suction at the other end, solid product quality was improved greatly and operation steadied. The other improvement was the introduction of simple dams and lifters in the retort to control the flow and retain material in process.[10]

The Wisner process as practiced at Philo and as engineered into the Champion pilot plant had a large rotary, star-shaped seal through which the solid product was discharged. The purpose was to prevent air from entering that end of the retort. This was a large cast-iron device moved from Philo to Champion. In the course of time it became worn and battered and was no longer even the semblance of a gas seal. However, the more the seal leaked, the better the operation of the carbonizer. This fundamental was not appreciated fully or accepted however, until the three large units comprising the first Disco plant had been built and operated for some time. Leaving the discharge end of the carbonizer open and by suction on the upper closed end, pulling the product gases together with air into the open end, solved some difficult problems of continuous carbonizing in a revolving retort. There was no necessity for a seal to withhold air and yet pass large pieces of solid product. The combustion of the oxygen developed heat where it was most needed, supplementing that which was transferred from heating gas through shell. Without the supplemental heat from combustion by open-end air, the solid product was soft and smoky. Open-end operation presumably burns tar, for the tar yield is much lower than with the closed-end operation. However, the discovery of the open-end made it possible to produce Disco with a marketable structure.

Control of the flow of material through the carbonizer by dams and lifters was another important development in the ball-making process that came from the operation of the three large units. Some aspects of open-end operation and dams and lifters are covered later in this paper, but for more detailed discussion of these factors and the theory of ball making, the reader is referred to the earlier paper on the Disco process.[9]

Seven Unit Plant

The Pittsburgh Consolidation Coal Co. in March 1947 authorized the construction of a new Disco plant to carbonize 1000 tons of coal per day. The first unit was in production in April 1949 and the plant in full operation in December. The coal, which contained 13 pct moisture, was to be delivered by belt from the Champion preparation plant of the Pittsburgh Coal Co. to a storage pile from which it would be withdrawn currently, heat dried to 3 to 4 pct moisture and processed to Disco, tar, and gas. Seven process units were specified, each to take 150 tons of coal in 24 hr. The new plant site adjoined the original three-unit plant.

The record of more than 2 years' performance of the process units has been particularly gratifying. Unit input in the first year of full operation was 148 tons of dry coal per day, with a yield of over 75 pct commercial Disco and between 16 and 17 gal of tar per ton of dry coal. In 1951 the throughput was increased about 10 pct to 165 tons of coal per day. The yield has been maintained at 75 pct. At this rate of input the daily production of Disco is 860 tons as compared with the design figure of 770 tons.

Carbonizer

The carbonizing retort, its length, diameter, slope, speed of rotation, and its capacity, was the first consideration in design and layout of the new plant. The carbonizer in this process has the primary function of rolling plastic coal into balls. The retort is not an efficient heat transfer machine, and its capacity cannot be calculated from the theory of heat transfer. Given properly prepared feed, and adequate heat input to complete the carbonizing reactions, the capacity of the Disco retort is determined by volumetric considerations. The limiting factor is bed depth in the plastic zone.

Three stages are represented in the progress of material down the length of the carbonizer. The first is heating the coal and recycle breeze from about 600°F to its softening temperature. The second begins when the material softens into a plastic mass. The third begins when the plastic material has solidified sufficiently for individual pieces to retain their identity. In other words, a dry, a wet, and a dry stage. The size of the individual pieces, the balls, is entirely fortuitous in any given bed condition, with the largest balls resulting where the depth of plastic material is greatest.† The diameter of the retort also has its effect on ball size. Along any foot of horizontal length, the larger the radius of retort, the more volume at any given maximum depth, hence the more likely that large pieces will result when the postplastic stage is reached. This is evidenced by the larger proportion of maximum sized balls from the largest sized retorts. It had been found by experience with the 8-ft retorts that a 6-in. bed of coal and breeze was the deepest practical depth. This was controlled by the height of internal dams, 6 in. high, at intervals along the shell.

Experience gained in the operation of 3, 6, and 8-ft diam continuous retorts and 3, 8, 10, and 12-ft diam batch retorts had supplied the essential data for the design of the new retorts. A 10-ft batch carbonizer, originally 12-ft, is shown in Fig. 7. Slopes of ¼, 1/3, ⅜ and ½ in. to the foot had been tried on different retorts. Experimenting with the effect of varying speeds of rotation had demonstrated that a peripheral speed of 75 to 100 fpm was a good compromise between power consumption and effective

† In the earlier report on the Disco process, it was categorically stated that the size of the balls is determined by the agglutinating property of the coal when it has become plastic; and that structure is determined by the inerts as well as by time-temperature treatment. This is not inconsistent with the present assertion that depth of bed in the carbonizer is the controlling factor. One factor supplements the other. Other factors being equal, bed depth controls, but in any given depth of bed, the coal that has the greater agglutinating property will stick together in larger masses before solidifying than another less strongly coking coal. This has been demonstrated experimentally in the batch retort where the bed depth can be predetermined and other process conditions be controlled with the coal quality as the only variable.

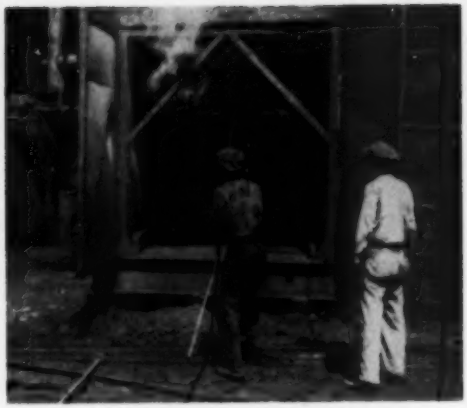

Fig. 7—Ten-foot batch carbonizer used for study of the effect of retort diameter on Disco ball formation.

mixing and conveying of the charge. The new retorts have a peripheral speed of 75 fpm and a slope of $\frac{3}{8}$ in. to the foot.

The hot gas seal at the upper end of a 9-ft carbonizer is shown in Fig. 8. Two steel rings forming the seal are indicated by the arrows. The ring at the left revolves with the retort. The ring at the right is anchored from revolving, but moves in and out on the axis of the retort. It is held in contact with the revolving ring by counterweights. To the left of the tire is the packed gland which is the seal for the byproduct gas. Both seals are under pressure of a few millimeters. The steel wear strips on the hot gas seals are lubricated by graphite plugs in the metal and frequent applications of flake graphite.

The gas offtake end of the carbonizer is shown in Fig. 9. The large elbow duct in the center leads from the upper cone end of the retort into the cast-iron dust box in lower center. Tar vapor and gas pass down onto a water seal in this box, depositing the greater part of their dust burden, then to the right and up through the water scrubber at extreme right. The conveyor in lower left removes dust from water in the center box. In the upper left is the carbonizer feed screw leading into the retort through the side of the gas elbow. The charge enters the screw through a vertical chute from the roaster-storage feeder on the floor above. The next adjacent carbonizer shows in left background.

It is customary to relate the capacity of the Disco retort to the tons of coal processed in 24 hr. This is misleading, however, because the total heat load and processing requirement is the coal plus the recycle breeze. In the first plant the ratio of coal to breeze had varied, by units, from 25 to 30 pct breeze, with a general average of 30 pct for the larger units. That is, for 70 parts of coal charged, there were 30 parts of breeze. Tests had been made in which the ratio of coal to breeze was 60 to 40, and with improved conditions as to preheat temperatures, it was decided that it was practical to design the new plant for such a ratio. It was decided further to preheat breeze and coal together in the roaster storage system, instead of separately.

To process 1000 tons of coal per day and recycle 670 tons of breeze would require eight of the 8-ft retorts, or seven, of 9-ft diam. The 9-ft retorts were specified, each rated at 150 tons of dry coal and 100 tons of recycle breeze, a total load of 250 tons per day, or more than 10 tons per hr.

The length of the new carbonizers was specified as identical with original No. 3, 129 ft, cold. With center lines of heating gas hoods 92 ft $9\frac{3}{4}$ in. apart, the effective heating area was 3016 sq ft. The original unit, with 6-ft diam was 76 ft between centers of heating gas hoods, the second was 8 ft in diam but with the same length. The third had the same diameter as the second but was 20 ft longer. Operating experience over a period of years with substantially the same heating gas temperatures and the same feed conditions showed that an increase in diameter had more effect on capacity than an increase in length. Thus with the two retorts of the same length, the 8 ft, with an increase in heating area of 25 pct, had 37.5 pct greater capacity than the 6 ft. Comparing the two 8-ft retorts, the longer, with an increase of 29 pct in heating area, gained but 18 pct in capacity. The new units, 9 ft in diam, have 13 pct more heating area than old No. 3, and were designed to carbonize 15 pct more coal. As will be explained subsequently, the new units are handling 23 to 25 pct more coal than the largest older 8-ft unit.

The minimum necessary length of the carbonizing retort is that required for heating the preheated coal to and through the plastic range. After the charge has hardened sufficiently to assume the final shape and size of product, longer retention in the revolving retorts serves only to devolatilize it further and to grind it as in a ball mill. One result is desirable; the other is not. Experimental evidence of the reactions in the continuous carbonizer is confined to observations of the elapsed time for predetermined changes in feed to be noted in product discharge and to inspections of the cold interiors of retorts after a shutdown. The time of transit through the retort is approximately 1 hr. Additional and highly informative data are available from studies in batch carbonizers in which temperatures were controlled and measurable, and in the operation of which a test was terminated at any desired stage of heating or carbonization and the products examined and analyzed.

From consideration of all the data thus available, it is estimated that charge entering the 126-ft continuous carbonizer at 600°F travels about 45 to 50 ft in 25 to 30 min before it becomes plastic. In this time it has been heated to nearly 800°F at an average rate of 8°F per min. In the next 15 min the coal has passed through the plastic range and with the recycle breeze has become consolidated into irregular-shaped masses not yet hardened. In the next 10 min the material has traveled another 25 ft, has increased in temperature to about 875°F, and has become hard and firm.

There is on the discharge end of the retort what is termed the cold extension, extending about 20 ft beyond the tire and the heating furnace. In this extension the balls are retained for about 10 min for further devolatilizing and cooling before being discharged at about 800°F.

The conditions described above were set by experimental work on the three retorts in which, in the first few years of operation, dams and lifters were raised repeatedly, lowered, and moved to affect the flow and heating of charge to attain the maxi-

mum desired effects. Retention time in the shorter retorts was maintained substantially the same as in the longer one by slower feed rates, not by deeper beds.

It has been noted previously that the new 9-ft retorts handle a much greater tonnage than the old 8-ft, No. 3 of the same length. The input tonnage, coal and recycle breeze, to the 8 ft was from 185 to 200 tons in 24 hr. The seven new units each take over 300 tons in 24 hr. The overall increase in capacity—in excess of that due to greater heating area—arises from better design of heating gas seals, more effective insulation, both conserving heat and in large part to the higher preheat of charge. In the old plant the average temperature of coal to carbonizer was 540°F and of recycle breeze, heated independently, 435°F. In the new plant the coal and the major portion of recycle breeze are heated together and delivered to the carbonizer feed screw at from 580°F to 610°F, with the remainder of the breeze charged simultaneously at 700°F or higher.

Roaster-Storage System

Three fourths of the heat required to make Disco is that necessary to dry and raise the charge material to 600°F for delivery to the carbonizing retort. This is done in the roaster-storage system by indirect heating.

Design data for the new roasters were derived from a study of the performance records of the two types used in the old plant, supplemented by analysis of heat balance data. No practical design of circular multiple hearth roaster gave the required capacity for the new units. The straight line rectangular hearth, originally developed for the first unit in 1933, was extended and widened for the new plant. The design called for 2100 sq ft of roaster hearth per unit, on four hearths 6 ft wide and 90 ft long, with rabbles every 3 ft, travelling at 30 ft per min. The bucket conveyor on the lower two decks, the storage, carries around; that is, material not taken from its discharge by the carbonizer vane feeder is lifted to the upper deck and makes another trip around. Because the carry-around capacity is limited, fine adjustment is required between feed rate to roaster and storage discharge rate. Charge material is in the four decks of roaster about 40 min, and in storage conveyor from 10 to 30 min depending upon the load.

The normal rate of temperature increase in process is from 150°F as charged to 435° to 475°F off the second roaster deck; 540°F to 560°F off the fourth deck and 580° to 610° at the final discharge from the storage into the carbonizer. Air for oxidation of coal need be admitted only to the lowest or storage decks. It is of greatest importance to maintain the coal in this preheating equipment below the softening point, and of course to avoid fires on the decks. Experience has shown that a top temperature of 600°F of the roaster-storage discharge is safe; that if it strays above 640° trouble will result, for this hot material is highly reactive and will take *off*, coke or fire, or both. Temperatures at ten points in the system are recorded currently and constantly watched by the control room operators, as are the power loads on all heavy equipment.

The four roaster decks and the top storage deck rest on ducts through which flue gas returns from the upper end of the carbonizer to the recirculating fan and furnace without contacting the coal. Temperature of the heating gas drops from about 900°F

Fig. 8—The hot gas seal at the upper end of a 9-ft carbonizer. Two steel rings forming the seal are indicated by arrows.

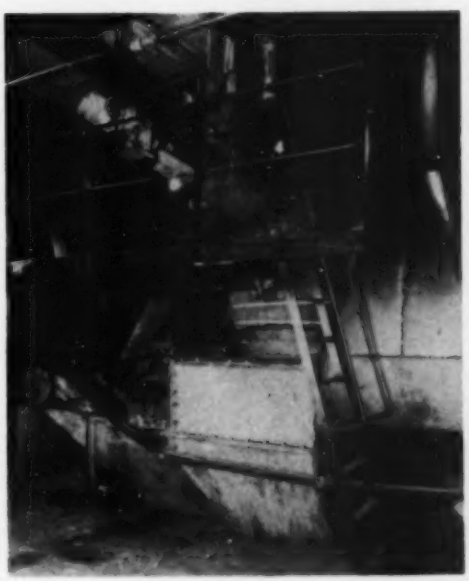

Fig. 9—The gas offtake end of the carbonizer. The elbow duct in the center leads from the retort into the dust box.

to 600°F in its passage through the roaster-storage duct system. Dampers in the inlet to each deck permit close control of temperatures. As designed, excess flue gas may be vented from a point between the fan and furnace to the second deck of the roaster, and in turn vented from the top deck to atmosphere through unit cyclone dust collectors. This hot, 600°F, sweep gas over the first two decks not only assists in initial heating but keeps the effluent gases above the dew point, below which serious corrosion of steel would result.

Serving the seven units is a distributing conveyor containing the desired proportions of coal and recycle breeze. From this common conveyor through a 1-hr supply bunker, a measured quantity is taken to the top deck of each roaster and after heating is delivered to the carbonizer feed screw. Each unit, roaster and carbonizer, is an individual process machine for which it is necessary to maintain uniform conditions of temperature and material flow. Changes in feed rate and temperature through the system are gradual; no quick reactions can be expected, but rather slow upward or downward trends. Effective and positive results, however, follow changes in the coal-breeze ratio to the carbonizer. The operator, who, by frequent visual inspections of the solid product from each unit, finds it necessary to alter the product, because either too small or too large in average size, has the means of correction at hand in the rate of flow of supplemental hot breeze directly to the carbonizer feed. The breeze for this circuit is that screened out through ½-in. round perforations in the nozzle at the discharge end of each retort. It is collected and elevated to a higher level in insulated conveyors for distribution at about 750°F. By remote control the operator manipulates the feeder and the flow of this hot breeze to each carbonizing retort. Approximately 25 pct of the total recycle breeze flows in this circuit, giving a marked improvement over previous design in heat utilization as well as in product control.

Operation

This is a continuous operation, designed with comparatively little leeway. From the stockpile, coal moves continuously to and through the heat drier, a single surge bin holding a 1-hr supply, a weightometer, and finally to individual bins above each roaster. From the stockpile, through the heat drier and roaster, and out the carbonizer as Disco, adds up to less than 4 hr in process. The heat drier, surge bin, and roaster feed bins normally afford 1 hr or more of time to recover from failure in this line of supply, and complete stoppage here would not introduce any hazard or serious operating problem.

However, from the carbonizer discharge to the cooling wharf, the line of flow is vulnerable because the material is hot and will take fire at any stoppage. No machine is infallible, particularly those handling hot, combustible material, hence every effort has been made to provide alternate routes around possible points of breakdown. The many provisions in the original design have been supplemented as experience indicated the necessity. The ability and resourcefulness of the experienced key operating personnel, both supervisory and hourly, has surmounted one problem after another until now the major deficiencies in the original design have been largely overcome.

It is fortunate for the economics of the Disco process that recirculation of fines is necessary for marketable product. Since the process requires all the fines incidental to the operation, there is no problem of breeze disposal. It has been noted that the old Disco plant charged 70 parts coal to 30 parts recycle breeze. Under normal conditions this worked out in balance when the breeze was all solid product through 1¼-in. screen.

The new plant was designed for 60:40 coal to breeze ratio, and as has been stated, is operating on approximately a 50:50 ratio. The difference between the two plants is in the amount of oxidation, which in the new plant is much less than in the old. There are two sources of breeze, normal degradation in the retorts and in the handling that follows through cooling to final screening, and degradation from rehandling from a stockpile, see Figs. 10 and 11. In the operation of the present plant, it has been found that when reclaiming a tonnage equal to production, sufficient breeze for recycle is available to allow the loading of +1½ to 1¼ in. Disco. When going into stock with current production, as in the summer months, it is necessary to crush everything below 2 in. to get sufficient breeze.

Retort Maintenance

Deposits on the inner walls of retorts plague every user of the rotary kiln in which material is heated until it becomes soft or plastic. If the end product is to be a liquid, or clinker or coke, the problem is the same. Charged dry, to emerge in any of these forms, the material must pass through a soft or sticky stage, and in the transformation, fused material clings to what it contacts, whether steel or refractory.

Distilling or carbonizing coking coal in a kiln or revolving retort by either direct or indirect heating has always been a troublesome operation by reason of build-up of carbonaceous accumulations on the walls. Innumerable attempts to remove such accretions as they formed by devices rolling around in the retort, cutting or chopping scale from the shell, have proved futile. The reduction of this problem by a combination of roasting the coal and admixture of inerts, as developed in the Disco process, has already been described.

However, there are in the Disco retort accretions that do not affect operations over long periods. Carbon from cracking of the tar vapor makes a hard, dense, slick coating that clings tightly to the steel. It is evenly deposited and shows every minute irregularity of the surface on which it is deposited, such as welds. Such a surface is shown in Fig. 12. This type of deposit is particularly noticeable in the upper part of the retort, above the region where coal becomes plastic. After months of operation this type of deposit will vary in thickness from less than ¼ in. at the feed end to ½ in. half way down the length of retort. This deposit does not greatly interfere with heat transfer through the shell, nor does its accumulation interfere with the orderly process of carbonization.

From the location where the coal becomes soft and sticky to the discharge end, the deposit over a period of months becomes much thicker than that nearer the feed end. It has a rough pitted surface, differing from the glassy surface of the first type described, and is very hard. After several months of continuous operation this hard deposit in the lower one half

or two thirds of the retort becomes thick enough to affect heat transfer and thus quality of product unless removed. Normal practice is to clean each retort twice a year. A unit is off the line for 7 days for cleaning with air-chipping tools.

In the immediate area where the coal softens, a porous deposit, often 2 or 3 in. thick, collects on the harder scale. This porous material breaks off intermittently and is discharged in small amounts with the Disco balls. The operators recognize the pieces as flats. For the most part, such pieces, when they fall off, are ground up by the balls before discharge.

A third type of deposit, the ring or doughnut was of regular occurrence in the three-retort plant. Concentric rings of a mixture of pitch, coke, and coal accumulated on the steel at the upper end of the retort. These rings, which grew to such heights as to obstruct the flow of incoming feed, were caused by condensation of pitch on the cooler charge as it flowed from the feed screw. No such ring accumulations have been found in the new units after more than 2 years of operation. The explanation of this improvement is that the charge, coal and breeze, is not only more carefully and intimately mixed, but that the charge as well as the steel retort itself are hotter in the new design and more uniform in temperature, hour by hour than in the old. The story of deposits in the carbonizers can be summarized by noting that although initially an outstanding problem in the development of the Disco retort, there is now no difficulty on this score.

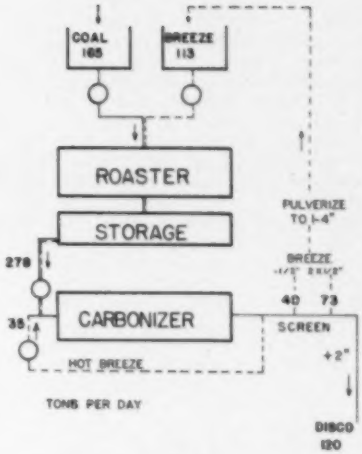

Fig. 10—Flow diagram of solids in one 9-ft Disco carbonizer unit, in tons per day, 165 tons of dry coal making 120 tons of +2-in. commercial product.

Physical Structure of Disco

The current measure of quality of Disco for commercial shipment is its shatter index. Usual testing procedure is daily determination of the shatter index, ASTM standard, on +2-in. sq hole screen. Normal operation gives a product off the cooling wharf ranging between 65 and 70 shatter index. The strength of the ball as measured by its shatter index is directly related to the rate at which it was cooled. Balls collected as they were discharged from the retort and tested while hot, have shown shatter tests of over 80 pct on 2 in. The same balls cooled suddenly, as by immersion in water, will be very fragile. Slow cooling in the air has been developed as the way to preserve the strength of the Disco ball.

There is another factor affecting the structure of the larger pieces. The larger balls are the result of soft masses having been rolled and kneaded in the retort. A mass of plastic material that is to become a ball is folded and rounded as it becomes solidified. The surfaces folded together are weak joints that may extend deep into the final pieces. This is illustrated in Figs. 13 and 14. The 10-lb ball of 800°F coke shows how this folding takes place. Smaller balls, 6 to 8 in. or less are initially more substantial than the largest balls.

These Disco balls were made in a batch retort in which 60 lb of 35 pct volatile matter coal and 40 lb of recycle breeze were charged for each test. Fig. 13 is a large ball taken from the retort when the charge temperature reached 800°F. The ball in Fig. 14 was taken from the retort when the charge temperature reached 900°F. The 800° ball was soft, plastic, and emitted tar vapor when taken from retort. If this ball had been retained in the revolving retort at increasing temperature, the crevices and cracks would have disappeared, and its appearance would have been that of the 900° ball. Moisture and ash free

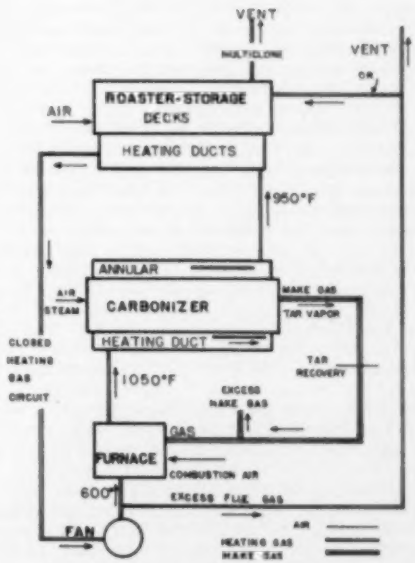

Fig. 11—Flow diagram of heating gas and of product gas and tar vapor in a unit of the Disco process. Heating gas does not contact the coal in process except in the open-end carbonizer.

Fig. 12—Interior wall of carbonizer showing accumulation of hard, pitch-like crust on every irregularity of the steel.

volatile content of the 800° ball was 22.0 pct and of the 900° ball, 18.4 pct.

Volatile Matter in Disco

The analysis of any coke will depend upon the coal from which it is made insofar as ash and sulphur are concerned. The volatile matter content will vary with the process, depending largely upon the temperature of carbonization. Volatile content of Disco is limited within a fairly definite range by two factors. One limitation is that imposed by the process which follows from the fact that once the plastic material has become the Disco ball, there is little subsequent change in analysis. The other factor is that the temperature of carbonization is limited by the safe working temperature of the metal in the retort. Disco carbonizing retorts have all been constructed of low carbon steel of flange quality for which the safe limit of working temperature has been considered to be 1112°F (600°C). Normal operation is with maximum heating gas temperatures of 1050°F and, of course, something less for the steel. The steel in the three original retorts, when dismantled and scrapped in 1950, after 12 to 15 years' operation under these conditions, exhibited no signs of deterioration or failure due to heat.

Solid product as it is discharged from the continuous retort ranges in size from large egg-shaped balls, 10 to 20 in. in length, to fine dust, the result of grinding. There are measurable differences in volatile content of different sizes. Because low temperature coke is a very poor heat conductor, the larger pieces are devolatilized but little after having been formed; the smaller sizes, and particularly the fine, −¼-in. material, is devolatilized further as it contacts the hottest metal of the retort. Test data have

Table I. Volatile Matter, Disco Compared with Fischer Retort

Temperature of Carbonization, °F	Fischer Retort		Disco Batch Retort	
	Dry	MAF*	Dry	MAF*
700	31.7	35.3		
750	24.8	27.9		
800	19.4	22.1	20.0	22.0
850	16.7	19.1	17.4	19.4
900	13.9	15.9	16.3	18.4
950	13.1	15.1	15.9	17.7
1000	10.3	11.9		

* Moisture and ash free.

shown an average of 0.6 pct lower volatile matter in the 1x2-in. size as compared with the large +2-in. balls, and a further drop of 0.5 pct from the 1x2-in. to the −⅜-in. breeze; a total difference of 1.1 pct.

Available evidence indicates that the balls are formed by the time the plastic material has reached 800°F. At that point the volatile content is about 20 pct, dry basis. The larger pieces lose volatile matter from that point on until finally cooled. From the data available, it is believed that the conditioning of the solid product in the continuous retort, that is, the retention in the retort after balls are formed, results in a reduction in volatile matter of 1 to 1½ pct, and that there is a further devolatilization of the large balls after discharge of a fraction of 1 pct, perhaps as much as 0.5 pct. The net result is that the larger pieces from the new 9-ft retorts, when cooled, analyze about 18 pct volatile matter. This strongly suggests that the actual temperature of carbonization in these retorts is between 825° and 850°F. Measurements of the temperatures at centers of medium sized balls as discharged give a consistent record of 800° to 810°F.

Significant correlations of the effect of time and temperature on the production of Disco balls are offered by test results in the 3-ft laboratory batch carbonizer. In this retort, weighed quantities of coal and recycle breeze were carbonized at predetermined rates of heating and to measured temperatures of the material in process. Table I gives the volatile matter analyses of a series of such tests compared with analyses of the carbonized products obtained in the Fischer retort. The charge to the Fischer retort was a few grams of coal alone; to the batch retort the charge was 60 lb of coal and 40 lb

Table II. Analysis of Byproduct Gas, a Composite of Gas from Seven Units as Fed to Burners

Component	Pct
Carbon dioxide	5.3
Hydrogen sulphide	0.9
Oxygen	0.1
Carbon monoxide	9.4
Hydrogen	3.4
Methane	10.9
Higher paraffins, as ethane, C_2H_6	8.9
Ethylene, C_2H_4	2.6
Propylene, C_3H_6	1.2
Butylene, C_4H_8	1.0
Nitrogen	56.3
	100.0
Calculated heating value	Gross 412.6 Btu/SCF*
	Net 380.2 Btu/SCF*
Calculated specific gravity	0.95
Quantity per ton of dry coal	3700 SCF*

* Standard cubic feet.

Table III. Typical Analysis of Disco Tar

Moisture, pct by volume	3.5
Specific gravity at 25°C, dry	1.15
Distillate to 270°C, wt pct	23.4
Ash, wt pct	1.7
Dry tar acids in tar, volume pct	12.5
Specific gravity of tar acids	1.03
Cresol, volume pct	8.0
Creosote oil in dry tar, volume pct	31.0
Phenol in dry tar acids, volume pct	10.0
Phenol in dry tar, volume pct less than	1.0

of recycle breeze. The analyses in this table are of balls about 9 lb each.

Byproduct Gas

Byproduct gas produced is more than sufficient for process heating. During the first few years of operation of the three-unit plant there was a surplus, but in later years insulation became impaired and the original unit, No. 1, with its four hot gas seals, was difficult to maintain, with the result that after 10 years' service, the plant had a deficit in available heat from make gas. As a further indication of inefficient use of gas in the old plant, it is noted that flue gas from No. 1 unit had 4 to 5 pct CO_2; from No. 2, 7 pct; but No. 3 was between 11 and 12 pct, as are the units in the new plant. Corresponding static pressures over the heating gas circuits were respectively 2.8 in., 2.2 in. and 1.8 in. for units 1, 2, and 3. Static pressures across the circulating hot gas fans are 2.2 in. of H_2O in the new plant. An analysis of byproduct gas is given in Table II.

The deficiencies were overcome in the design of the new plant by the best possible insulation, effective seals and gas tight expansion joints, with particular attention to pressures on the plus side of the rubbing seals. A considerable part of the unaccounted for heat loss, as shown in heat balance, is unavoidable leakage through the two rotating heating gas seals on each carbonizer.

Tar

Tar is the only byproduct presently available from the Disco operation and as such is important in the economics of the process. When the project of making low temperature coke was initiated, it was recognized that whereas the market for the solid product was obvious, that for tar was unknown and uncertain. It is true that many ambitious schemes for thus processing coal had been advanced on the

Table IV. Material Balance, Basis 1 Ton Coal, 4.2 Pct Moisture (For the Seven-Unit Plant)

Input	Lb Per Ton Coal
Dry coal	1916
Moisture in coal	84
Natural gas (pilot flames)	1
Air for combustion, dry	1069
Air to roasters, dry	740
Air to open ends of carbonizers, dry	189
Moisture in air	10
Steam entering open end of carbonizer	36
	4035
Output	
Disco	1302
Tar, dry	155
Gas, excess, dry	69
Moisture in tar and gas	6
Vent gas with entrained dust and moisture	2136
Liquor and losses	167
	4035

Fig. 13—Large ball taken from retort when the charge temperature reached 800°.

basis of exceptional values from the tar and oil byproducts, but none had then, nor have they since, materialized. The steps in the development of the manufacture of a smokeless lump fuel have been described, and it has been indicated that the effort was directed toward improving the solid product with little or no reference to the tar. However, the time came when it was necessary to find outlets for a tar that was a new and different material and for which there was no recognized market.

The problem of the tar thus was one of finding a market, and as this paper is concerned primarily with the process of making Disco by low temperature carbonization, the story of tar will be but briefly covered. From 1933 when the first tar was produced until the end of 1948 when the old plant was abandoned, nearly 16 million gal of Disco tar were produced and marketed in one form or another.

It is interesting to note that over 17 years of operation tar sales represented 24 pct of gross revenue and Disco 76 pct.

One ton of the 35 pct volatile coal from the Pittsburgh bed will yield 25 or more gal of tar when carbonized at 850°F to 900°F in a closed retort. The same coal carbonized in the open-end continuous Disco retort yields 15 to 17 gal of tar of essentially the same analysis. Table III is a typical analysis of Disco tar.

Marketing this tar met with many difficulties. None of the established high temperature coke oven

Fig. 14—Ball taken from retort when the charge temperature reached 900°.

Table V. Heat Balance on Disco Plant, Thousands of Btu

	Three-Unit Plant 1947		Seven-Unit Plant 1930	
	Btu/Ton Coal	Pct	Btu/Ton Coal	Pct
Input				
Gross heating of underfiring				
Byproduct gas	1577.9	68.0	1122.0	72.7
Natural gas	278.0	11.8	56.0	3.6
Open-end combustion	260.2	11.0	225.0	14.5
Roaster air combustion	72.9	3.0	65.0	4.2
Sub total	2189.0	93.8	1468.0	95.0
Sensible heat in coal	20.0	0.8	32.4	2.2
Sensible heat in steam	96.5	4.0	28.9	1.8
Sensible heat, water vapor in air	33.6	1.4	15.8	1.0
Sub total	150.1	6.2	77.1	5.0
Total	2339.1	100.0	1545.1	100.0
Output				
Sensible heat in solid product	525.0	22.5	606.4	39.3
Sensible and latent heat in tar	49.2	2.1	62.3	4.0
Sensible heat in dry byproduct gas	68.7	2.8	67.8	4.4
Sensible and latent heat in water, in gas, liquor and tar	280.0	12.0	214.0	13.8
Sensible heat in wash box refuse	0.5	0.2	1.7	0.1
Sensible and latent heat in moisture in stack gas	247.0	10.5	224.0	14.5
Sensible heat in dry stack gas	277.4	11.9	150.0	9.7
Radiation and convection	704.0	30.0	150.8	9.8
Not accounted for	187.3	8.0	68.1	4.4
Total	2339.1	100.0	1545.1	100.0

tar refiners wanted it. It was barred from road surfacing by specifications in many localities. In a region where steam coal was abundant and cheap, it could not be sold for fuel. The first sizeable outlet was as road tar, where by adjusting viscosity and by blending with purchased high temperature tar, a satisfactory road material was made and sold in quantity. Theoretically, however, the principal value should be in the low boiling, high tar acid content fraction; hence the next step was to refine the tar acid oil for sale as disinfectant oil. Wood preserving oils were then produced. During World War II the tar was refined into tar acid oil, creosote oil, fuel pitch and coke, all of which were marketed without difficulty. Subsequently, further refining was added and pure cresols were prepared for resin manufacture.

Fig. 15 shows the changes in Disco tar refining and market development over a period of years. During all this period the Disco organization designed and built the tar refining equipment, sold the products, and simultaneously did continuing research[14] on low temperature tar products. There remains much to be learned about such products, as for instance the high boiling tar acids, or the reduction of heavy pitches to high grade carbon.

Material Balance

The simplified material balance that follows, Table IV, and Figs. 10 and 11, recognizes coal and air as the materials in, and Disco, tar and make gas as the products. Recycle breeze does not appear in this record because the quantity used is equal to that made. Air into the system is not measured and the figures given are calculated, mainly from gas analyses. In Table IV the air to roaster is an approximation based on studies at the three-unit plant. The air admitted to roaster decks is appreciably less in the new plant than it was in the old plant. For neither, however, have reliable quantitative measurements of the air passing over the hot coal been possible; that is, of the quantity that could be expected to oxidize coal in the short interval of time it is exposed at temperatures above, for example, 300°F.

Heat Balance

There are three sources of heat in the Disco process: 1—Combustion of byproduct gas in the unit furnaces, 2—low grade oxidation of coal in the roaster, and 3—direct oxidation of solids and vapors in the carbonizer. A fourth source, of uncertain and undetermined magnitude is exothermic heat of carbonization in the carbonizer. Whether there is, or is not, exothermic heating during carbonization is a debatable point. Such possible heat has not been taken into any calculations of heat balance in this study. The three named sources of heat are, within limits, controllable; the fourth is not.

In the discussion that follows of heat developed and utilized in the Disco process, reference will be made to the three-unit plant and the seven-unit plant. It is important to note that the former comprised three units differing greatly in utilization of heat and that the data given are a weighted composite of the whole operation taken in 1947 after 12 to 15 years of use. The original No. 1 carbonizer was wasteful of heat, which explains the excessive use of supplemental heat from natural gas shown in Table V. No. 3 carbonizer gave the best record of the three as to tonnage and heat efficiency, and operating data from this unit were used in designs for the new plant.

In Table V are shown the input and output of heat for the new, compared with the old plant. Gross heat of underfiring, the heat from combustion of gas in the furnace, needs no explanation. The figures are for a given test period and are believed to be representative and accurate to within less than 10 pct. Data for the heat developed in the roaster decks by low grade oxidation and in the carbonizers by combustion are necessarily derived from studies of gas analyses, together with measurements of combustion air and gas involved.

It was estimated that 36 Btu per lb of coal resulted from oxidation in the roasters of the old plant,

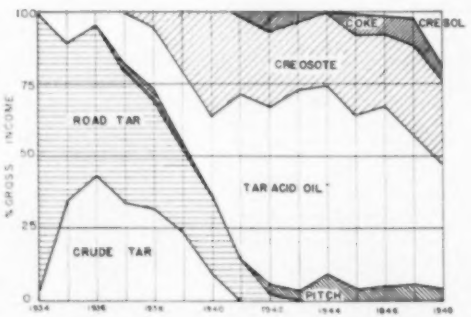

Fig. 15—Progress in development of refining and marketing of low temperature tar from the Disco process. Percentage of gross income from tar products from 1934 to 1948 of the operation of the original three-unit plant.

and a smaller quantity in the new plant where there is less dependence than in the earlier operation on reduction of coking property or more particularly on boosting coal temperatures in the roasting step.

Heat developed in the carbonizer by combustion of air pulled in the open end is a very important item in making Disco. Here again, the quantity of heat released is calculated from considerations of quantity and analysis of byproduct gas. Composition of the gas varies with the suction on the retort, operating temperatures, and the rate of coal feed. The composition and quantity of byproduct gas may be quite variable, and because proper working of the process depends upon uniformity of conditions, the control of suction in each retort is a matter for constant attention by the operators.

Stack losses of less than 25 pct are shown in Table V. All vented gases pass through the roaster dust collectors; flue gas from the furnaces over the top decks; roaster air direct; and open-end air into the byproduct gas, thence to furnaces and into flue gases. Waste gas is vented at temperatures ranging from 170° to 250°F; an average of 210°F was used in these calculations.

Heat was conserved in the new plant by better insulation and improved gas seals. In the newer plant there are, with minor exceptions, no packed stationary gas seals. Bellows-type metal connections replaced seals at points where it was necessary to take care of expansion. The rubbing seals at each end of the rotating carbonizers were redesigned for greater accessibility. A material saving in heat was realized by returning recycle breeze at high temperatures, as has been described.

From Table VI, which shows gross heat balance for the seven-unit plant, it can be calculated that the operation has a cold thermal efficiency of 85.5 pct; which compares with 74.6 pct for the three-unit plant. This figure is the ratio of heating value of products recovered, Disco, tar and surplus gas, 21,623,000 Btu, to heating value of coal and natural gas input, 25,256,600 Btu. Greater yields of Disco, tar, and gas contribute to this improvement.

Table VI. Gross Heat Balance, Seven-Unit Disco Plant, Basis 1 Ton of 4½ Pct Moisture Coal

	Thousands of Btu
Input	
Heating value of coal	25,200.0
Heating value of natural gas in pilots	56.5
Heat content of coal, and of process air and steam	77.1
	25,333.6
Output	
Heating value Disco	18,680.0
Heating value tar	2,570.0
Heating value excess gas	373.0
Heating value recovered dust	160.5
Heating value wasted dust	864.0
Radiation and convection	199.3
Heat content of products	1,269.8
Not accounted for	1,217.0
	25,333.6

Acknowledgments

Many contributed to the development of the Disco process as it has been described. It is not possible here to name all those whose unremitting efforts and intelligent research contributed to the success of the project, but special credit is due J. B. Goode, who has been in charge of operation from the beginning, A. A. Archer, Chief Engineer, R. A. Zimmerman and L. A. Winter, for conducting the research, and Caleb Davies, Jr., to whom, with C. C. Clark, is given credit for the development of the refining and the uses of the low temperature tar. Heat balance calculations were by David Kindler and Frank Crowe, Jr.

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Primary Blasting Practice at Chuquicamata

by Glenn S. Wyman

CHUQUICAMATA, located in northern Chile in the Province of Antofagasta, is on the western slope of the Andes at an elevation of 9500 ft. Because of its position on the eastern edge of the Atacama Desert, the climate is extremely arid with practically no precipitation, either rain or snow.

All primary blasting in the open-pit mine at Chuquicamata is done by the churn drill, blasthole method. Since 1915, when the first tonnages of importance were removed from the open pit, there have been many changes in the blasting practice, but no clear-cut rules of method and procedure have been devised for application to the mine as a whole. One general fact stands out: both the ore and waste rock at Chuquicamata are difficult to break satisfactorily for the most efficient operation of power shovels. Numerous experiments have been made in an effort to improve the breakage and thereby increase the shovel efficiency. Holes of different diameter have been drilled, the length of toe and spacing of holes have been varied, and several types of explosives have been used.

Early blasting was done by the tunnel method. The banks were high, generally 30 m, requiring the use of large charges of black powder, detonated by electric blasting caps. Large tonnages were broken at comparatively low cost, but the method left such a large proportion of oversize material for secondary blasting that satisfactory shovel operation was practically impossible. Railroad-type steam and electric shovels then in service proved unequal to the task of efficiently handling the large proportion of oversize material produced. The clean-up of high banks proved to be dangerous and expensive as large quantities of explosive were consumed in dressing these banks, and from time to time the shovels were damaged by rock slides. As early as 1923 the high benches were divided, and a standard height of 12 m was selected for the development of new benches.

The recently acquired Bucyrus-Erie 550-B shovel, with its greater radius of operation compared to the Bucyrus-Erie 320-B formerly used for bench development, allowed the bench height to be increased to 16 m.

Churn drill, blasthole shooting proved to be successful, and tunnel blasts were limited to certain locations where development existed or natural ground conditions made the method more attractive than the use of churn drill holes.

Liquid oxygen explosive and black powder were used along with dynamite of various grades in blasthole loading up to early 1937. Liquid oxygen and black powder were discontinued because they were more difficult to handle due to their sensitivity to fire or sparks in the extremely dry climate. At present ammonium nitrate dynamite is favored because of its superior handling qualities and its adaptability to the dry condition found in 90 pct of the mine. In wet holes, which are found only in the lowest bench of the pit and account for the remaining 10 pct of the ground to be broken, Nitramon in 8x24-in. cans, or ammonium nitrate dynamite packed in 6x24-in. paper cartridges, is being used. This latter explosive, which is protected by a special antiwetting agent that makes the cartridges resistant to water for about 24 hr, currently is considered the best available for the work and is preferred over Nitramon.

Early churn drill hole shots detonated by electric blasting caps, one in each hole, gave trouble because of misfires caused by the improper balance of resistance in the electrical circuits. Primarily, it was of vital importance to effect an absolute balance of resistance in these circuits, the undertaking and completion of which invariably caused delays in the shooting schedule. Misfires resulting from the improper balance of electrical circuits, or from any other cause, were extremely hazardous, since holes had to be unloaded or fired by the insertion of another detonator. The advent of cordeau, later followed by primacord, corrected this particular difficulty and therefore reduced the possibility of missed holes.

After much experimentation, the blasting practice evolved into single row, multihole shots, with the holes spaced 4.5 to 5 m center to center in a row 7.5 to 8 m back from the toe. Such shots were fired from either end by electric blasting caps attached to the main trunk lines of cordeau or primacord. The detonating speed of cordeau or primacord gave the practical effect of firing all holes instantaneously.

Double row and multirow blasts, fired instantaneously with cordeau or primacord, proved to be unsatisfactory in the type of rock found at Chuquica-

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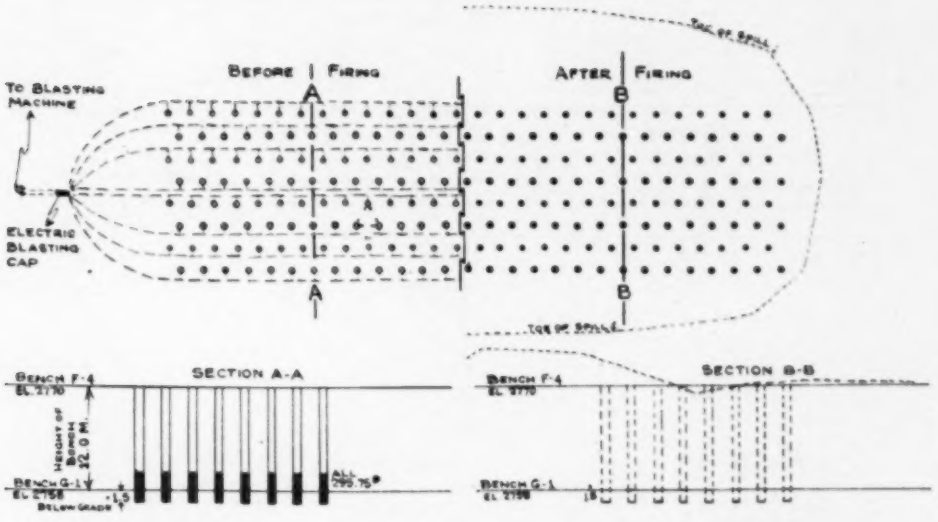


mata. The relatively high powder factor required to give anything like suitable breakage caused excessive back break, which in turn made drilling for subsequent shots difficult and in some cases made a satisfactory length of toe impossible to obtain on the next shot drilled. In cases where holes were drilled with an excessive toe, hard bottom invariably developed. Toe holes shot with the main blast were an expensive expedient used to correct the condition. Double row as well as multirow shots generally produced a large proportion of oversize material, which in turn caused excessive secondary blasting and poor shovel efficiency.

Loading sheets for all blasts were made up in the engineering office. These generally called for a large bottom charge extending from the bottom of the hole (1 m below grade) to a short distance above grade. One or more deck charges of smaller size were placed in the barrel of the hole. Hole size varied greatly, but in general it can be said that the comparatively small 6-in. diam holes used in earlier blasting were replaced by larger holes to eliminate insofar as possible the need for springing pockets at the bottom of holes.

New benches were developed by thoro-cut shots, also known as box-cut shots, 100 m long by 21.8 m wide. The blastholes were spaced 3.6 m center to center in rows 3.12 m apart, and the holes were staggered in the rows, thus giving an equilateral triangle pattern of 3.6 m a side. A shot of these dimensions required approximately 210 churn drill holes, all of which were fired instantaneously, using primacord throughout, see Fig. 1. These large shots were detonated from two points at either end of the center row trunk lines. The quality of breakage in thoro-cut shots was generally less than could be desired for the best shovel operating conditions, and back break, along with swell over the top rim, caused extra track work because a shovel cut was required around the entire area to allow for drilling the subsequent side shots. The first side shots were drilled before the broken material was loaded to obtain reasonable toe length, but this generally did not allow for drilling at short enough toe distance to

Shot No. 23 G-1 (Eight rows)			
No. of holes	224	Between rows, m	3.12
Length of shot, m	99x21.9	Drilled below grade, m	1.5
Diam of holes, in.	9⅞	Tons broken	111,747
Toe, m	none	Tons broken per ft drilled	11.44
Spacing, m	3.6 m equilateral	Tons broken per lb explosive	1.69

Fig. 1—Eight rows of churn drill holes all fired instantaneously, using primacord throughout.

give good breakage in these shots so that both thoro-cut and first side shot tonnage had to be loaded under poor conditions.

During 1949 considerable revision of shot layout was made because of the adaptation of the *short toe wide space between holes* pattern of drilling and the use of electric shot timers. A great improvement in fragmentation, as well as less back break resulted from this change. For example, it has been found that rock, which in the past would have been drilled to an 8-m toe with 5-m space between holes, can be broken to better advantage using a 7-m toe and 9-m space between holes with an increase in tonnage broken per foot of churn drilling and at no increase in explosive factor.

In recent years manufacturers of electric blasting caps have developed millisecond delay caps. The use of these caps permits a close rotation, or sequence, in firing. To avoid the necessity of carrying many different interval caps in stock, and also to decrease any probable differences in the time interval inherent in the caps as manufactured, a machine, or timer has been developed that permits blasting with millisecond interval delays when using the ordinary electric blasting caps.

Timers make multirow shots possible with no sacrifice in quality or breakage and with no increase in explosive factor, and at the same time they have the advantageous effect of limiting throw. By the use of timers it has been possible to lay down double row shots in the same bench area formerly required for single row blasts. This gives an advantage to the power shovel because the height of cut during

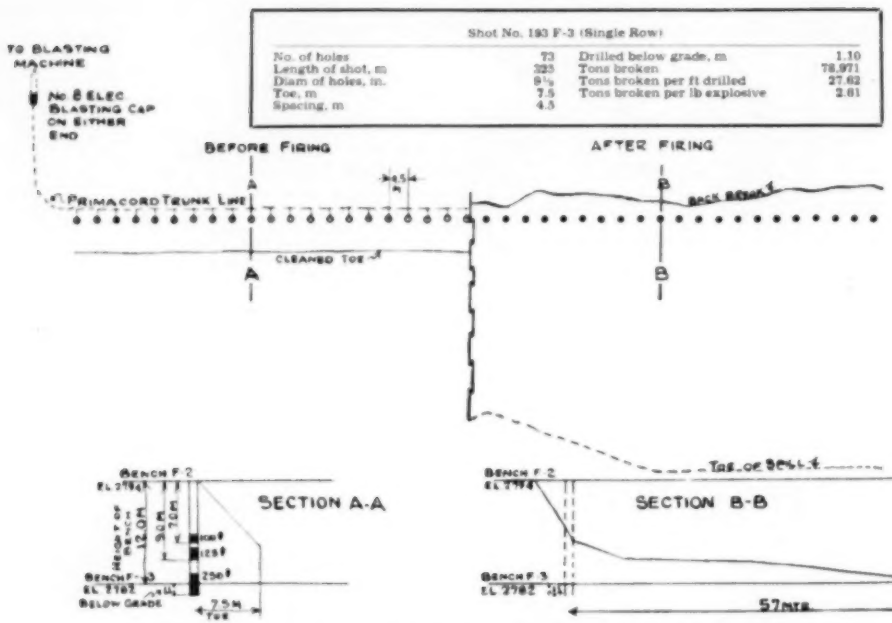


Fig. 2—Single row shot, which breaks about 28 tons per ft of hole drilled.

the first pass is greater. The tons loaded for each position of the loading track are increased.

The double row shots fired using four timed circuits have numerous advantages over the single row closely spaced type of shots fired instantaneously. There is a saving in churn drill footage since the single row shot, shown in Fig. 2, breaks about 28 tons per ft of hole drilled while the double row shot, due to wider spacing of the holes, breaks about 47 tons per ft of hole. It will be seen from a comparison between Sections BB of Figs. 2 and 3 that the throw of the single row shot is about the same as the throw of the double row shot measured from the back row, and it is obvious that shovel loading conditions will be superior in the higher cut obtained by the double row shot provided that fragmentation in the two types of shot is equal.

The dynamite factor used in the particular shots shown in Figs. 2 and 3 is actually more favorable in the case of the double row shot, being 2.61 tons broken per lb for the single row compared to 3.16 tons broken per lb for the double row. In general, it has been possible to improve the explosive factor through the use of the short-toe, wide-space, timed circuit type of blasting. Without doubt, some of the possible gain in cost per ton broken has not been exploited to the fullest degree because an attempt is being made to improve overall breakage and thereby improve overall efficiencies of shovel operations. The point at which savings in first cost of breaking ground become real gains through savings in the amount of explosive used is a delicate one. Apparent savings can be dissipated easily in indirect costs brought about by the inferior quality of the broken ground. Such indirect costs are increased secondary blasting, repairs to shovels, and preparation for subsequent blasts in areas where high bottom must be removed before a succeeding shot can be fired. Experiments with explosive factors must be made in locations where changes in the quality of breakage will not have a direct effect on shovel production and at times when total mine tonnages are not of paramount importance.

A single failure in an ore area may have a direct effect on copper production and this loss, coupled with the cost of extra labor and material expended in handling poorly broken ground with power shovels, can overbalance savings made through reduced explosive cost.

To insure against misfires due to broken or cut primacord trunk lines, good practice has dictated the use of electric blasting caps at both ends of each line. In adhering to this procedure while using a single timer, certain difficulties were encountered. For example, in firing a double row shot, alternate holes of each of the two rows were connected to separate circuits by employing four primacord trunk lines. The average double row blast is 350 m long, consequently the use of duplex electric leading wire the full length of the shot for each circuit, plus damage caused to these leading wires by flying rock, proved both costly and time consuming. The complicated electrical connections invited errors because it was difficult to check circuits between the ends of long shots.

Current practice at Chuquicamata calls for the use of two synchronized timers, one placed at either end of the shot to be fired. Power for both timers is supplied simultaneously from a common source through one duplex leading wire running the entire length of the shot. The circuits from the holes to the synchronized timers thus are shortened and consequently are easier to check. This makes for less chance of error when connecting up and thereby decreases the possibility of misfires. To insure perfect results when using more than one timer in the

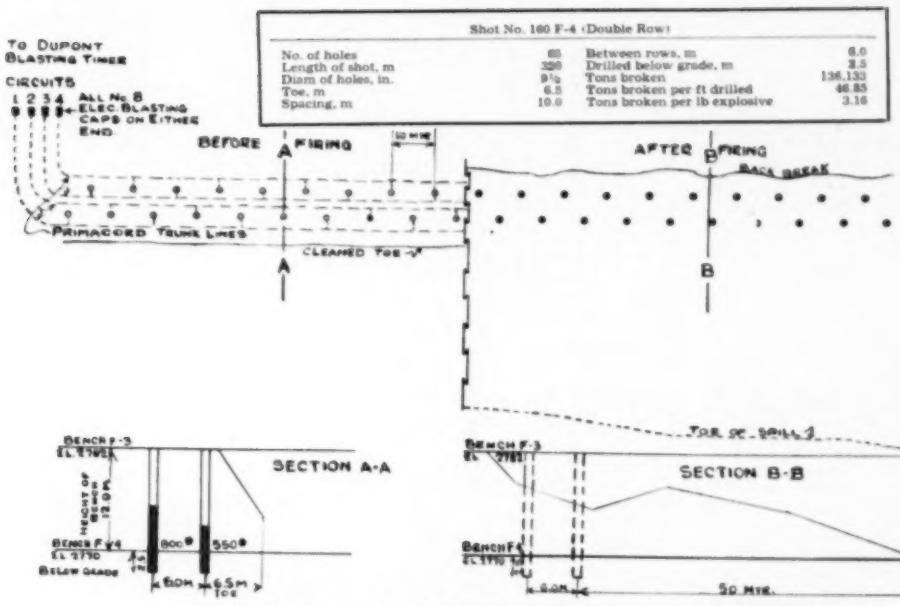


Fig. 3—Double row shot, which breaks about 47 tons per ft of hole drilled.

same shot, it is necessary to check the synchronization of the timers so that there will be no possibility of any time lag between the two units.

A great advantage is gained by shooting multirow shots, three and four rows, in areas that are to be worked by motor truck haulage, because sufficient floor space for turning the large 30-ton units is achieved with the first shot on the bench. On benches where the operations have been such that floor space had to be sacrificed or reduced to a minimum and subsequently must be regained, multirow shooting reduces the amount of overcasting to be done by the shovels in providing space for laying the railroad track.

Thoro-cut shots are made in the solid floor of the bottom of the pit to develop new benches. After the first 150-m block is broken by shots drilled to a pattern similar to that described above, the cut is advanced by smaller shots consisting of four rows of holes in which the holes are spaced 7 m center to center and the rows are 7 m apart. These shots are fired in conjunction with side shots in which the holes are drilled 10 m center to center and 9 m back from the toe, see Fig. 4. This has been made possible by the timers and has resulted in better broken ground with a consequent reduction in cost due to a greater number of tons broken per foot of hole and a greater number of tons broken per pound of explosive.

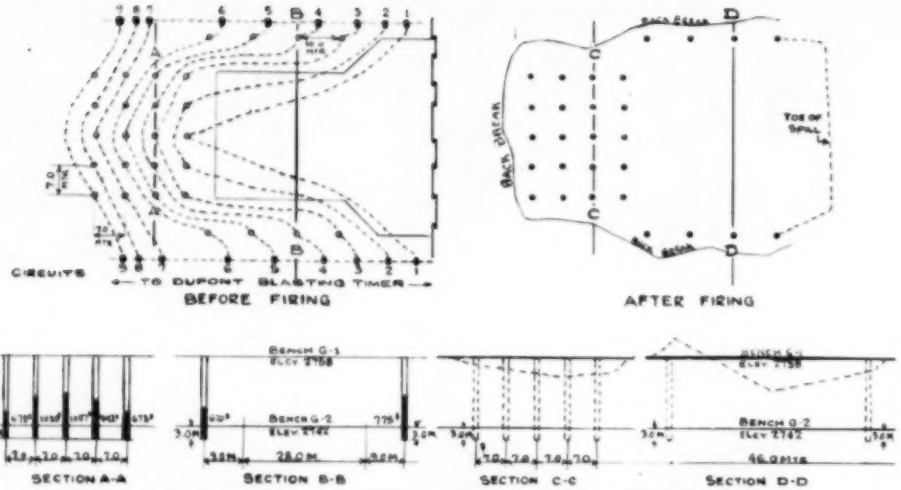
The advance of a thoro-cut by means of the old system where the holes were spaced close together required considerable preparation before drilling could be started, since each shot had a tendency to swell and spill over on the sides, as well as at the ends. This spillage had to be cleaned off before side shots could be drilled and before loading tracks could be put in position.

With the new type of blasting, four rows at a time, there is a limited back break and a minimum amount of work is required to permit drilling the side shots and placing the loading track in position. Fragmentation is improved greatly, thus increasing the shovel loading rate and decreasing the amount of secondary blasting required. Figs. 1 and 4 show the two types of shot and also indicate the order in which the circuits are connected to the timers.

Fig. 2 shows a representative single row blast fired instantaneously, in which the holes are drilled 4.5 m center to center in a row 7.5 m back from the toe. All holes are drilled 1.1 m below grade.

Fig. 3 shows a double row blast fired by the use of four timed circuits. In this case the holes are drilled 10 m center to center, in a row 6.5 m back from the toe. The second row is drilled 6 m back of the first, and all holes extend 2.5 m below grade. From a comparison of Section AA on Figs. 2 and 3, it will be noted that deck charges used in the single row shot are eliminated in the double row shot. This is because the charge per hole, due to the greater burden, is large enough to rise well up into the hole and thus take the place of the deck charges. This feature simplifies the loading operation in the field and reduces labor on the shot.

At present, preparations are being made to allow for drilling a large percentage of the total footage in 12-in. diam holes, rather than 9½-in. diam, which is the size now generally in use. As ground in different sections of the mine varies in breaking characteristics, experiments with even wider spacings than the 10 m shown on Fig. 3 are being made. A shot on Bench B-1 is being drilled to an 8.5-m toe with 12-m space between holes and 8 m between rows. The 12-in. diam holes will take the comparatively large charges required by this wide spacing without excessive rise of the explosive in the column of the holes. From previous blasting experience in this area good breakage can be expected. The breakage per foot of churn drill hole will be raised from 47 tons for a double row shot as shown on Fig. 3 to 61 tons at this increased spacing. As



Shot No. 20 G-2 (Four Rows)			Shot No. 24 G-2 (One Row)		
	20G-2"A"	24G-2"A"		20G-2"A"	24G-2"A"
No. of holes	20	8	Between rows, m	7.0	—
Length of shot, m	28x28	30	Drilled below grade, m	3.0	3.0
Diam of holes, in.	9½	9½	Tons broken	39,034	25,737
Toe, m	7.0	9.0	Tons broken per ft drilled	33.70	51.98
Spacing, m	7.0	10.0	Tons broken per lb explosive	2.33	3.62

Fig. 4—Four row shot fired in conjunction with side shots.

spacing is widened, the charge for a given churn drill hole becomes greater so that larger diameter holes are required to give suitable concentration at the bottom of the hole and to keep the column of explosive within reasonable height limits.

Experience tends to show that changes in length of toe produce a direct effect on the fragmentation, tendency to leave hard bottom, and general quality of breakage, a great deal quicker than changes in width of spacing. In other words, an increase of ½ m in the length of toe, see Figs. 1-4, shows detrimental effects from several viewpoints such as high bottom, excessive back break and general blockiness of the material broken while a change in spacing between holes of as much as 1 m in any given type of rock has shown no detrimental effect. The indication seems to be that there is a possibility of using even wider spacing between holes without sacrificing the quality of breakage and thus further increasing the tons broken per foot of churn drill hole. In hard and medium hard sections of the mine the size of the explosive charge that could be loaded in a 9½-in. churn drill hole limited the width to which spacings could be extended. Drilling with 12-in. tools will allow continued experimentation with hole spacings.

The successful substitution of paper casing for steel casing in the collars of the holes has resulted in a considerable saving. Experiments with quick-setting cement around the starter pipe indicate that there may be great possibilities in the use of this material and may result in holes being protected at even lower cost than is now done with paper casing. In certain sections of the mine raveling ground will require the use of some casing, but there is a possibility of lower costs by reducing casing consumption.

There is a question as to the most suitable delay period between circuits for the different shot layouts. The large majority of blasts fired at Chuquicamata with the electric timers have been with delays of 0.018 sec between circuits. Several also have been shot at 0.024-sec delay. The higher delays are probably more desirable as an aid in starting the holes of the first circuit in motion, thus furnishing relief for the holes of the next circuit. In formations free from cracking there seems to be little doubt that the higher delay periods will give the best results. The danger of cutoff holes becomes great in cracked or bedded formations, and while practically no missed holes have been encountered when using 0.018-sec delay period, there have been some holes cut off in shots where 0.024-sec delay periods were used. The electric timers are powered by 60-cycle motors and designed to give 0.010, 0.015, 0.020 and 0.025-sec delay intervals. Power in use in Chuquicamata is 50 cycle, and therefore the intervals mentioned above are changed accordingly.

The use of the timed circuit type of blasting becomes a separate study for each type of rock to be broken. There is little or no chance of cutoff holes in areas of little bedding or with few slips in the formation at any of the delay periods possible with the timing machine. In certain areas of the pit the rock is more difficult to break than in others, and in these areas slips and bedding are usually more pronounced. Therefore, the higher delay intervals are not considered practical, and while the higher delay intervals would no doubt produce better broken ground, they are not used because of the possibility of cutoffs and consequent misfires.

In conclusion, it can be stated that great advantage has resulted from the use of the short-toe, wide-space, churn drill hole pattern of shot layout in double or multirow blasts that are made practical by the millisecond delay intervals available with the electric motor-driven millisecond timing machines. There is still room for continued experimentation with the promise of improved quality of breakage and lower costs in Chuquicamata.

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Arizona Section Holds Annual Meeting at Tucson

THE annual meeting of the Arizona Section was held at the Pioneer Hotel in Tucson on Dec. 10, 1951. Registration and ticket sales for the social functions started at 8:00 am and lasted throughout most of the day. There were 489 registrations, consisting of 216 members, 64 students, and 209 guests.

Chairman Harrison Schmitt opened the Mining Geology Div. meeting and a short business session was held at which Dr. Charles Anderson of Prescott, Ariz. was elected chairman and Seton Williams of Tucson was elected secretary for 1952.

Dr. B. S. Butler was program chairman with R. B. Mulchay as co-chairman at the technical program. Eldred Wilson spoke on *Oxidation and Water Levels at Some Arizona Mines*. George Kiersch spoke about *Economic and Engineering Significance of Ground Water Localization in Limestones — Big Bend Country, Texas*. Harrison Schmitt gave *The Definition and Use of Terms in Mining Geology*. These talks were followed by discussions.

E. N. Pennebaker, A. D. Wandke, and H. Schmitt gave informal talks on foreign trips made during 1951, the meeting was then adjourned.

The Underground Mining Div. meeting was called to order by W. P. Crawford, Chairman. Andrew Robb was elected chairman for 1952.

John H. Jett, contract engineer for Phelps Dodge at Bisbee, presented a paper on *Material Handling at the Copper Queen Branch, Phelps Dodge Corp., Bisbee, Ariz.* which particularly covered timber packaging. R. U. Jackson of Hewitt-Robins discussed belt conveyors as applied to underground coal mining and calculations involved in determining the size of conveyor installations.

Chairman John A. Lentz, Jr. called the Open Pit Mining Div. meeting to order and the technical sessions began with Furman Byars presenting a paper on *A Brief Description of Bottom Dump Haulage*. Felix Berra talked on the *Ground Protective System of the Morenci Open-Pit Mine*. J. F. Gray discussed safety at Castle Dome and Copper Cities pits; R. W. Schilling gave an abstract of W. F. Sullivan's paper *Safety Procedure at the Chino Open-Pit;* and V. O. Wetzel talked about accident prevention at the New Cornelia Branch. K. R. Crocker of Cananea and J. H. Cazier of Bagdad talked on safety programs at open-pit mines of the companies.

At the business meeting, J. R. Watts of Inspiration was elected chairman for 1952.

The Ore Dressing Div. meeting was an all-day affair starting with an assembly of members at the College of Mines. The morning technical session consisted of a *Description of Continuous Pilot Plant at the College of Mines* by George Roseveare, metallurgist, Arizona Bureau of Mines, Tucson. This pilot plant has been used during the past two years in developing the metallurgical procedure for two important Arizona ores. The next item was *Demonstration of the Pilot Plant Operation with Discussion of Operating Experiences with the Plant*, by F. T. Davis, metallurgist, Magma Copper Co. and A. B. Romney, American Smelting & Refining Co.

After luncheon in the new student Union building at the University, the afternoon technical session began with a *Description of Equipment at the College of Mines of Interest to the Mill Man*, by John Anthony, College of Mines. The equipment described included the spectrograph, X-ray, polished and thin section equipment. Examples of applications to milling were presented.

Following the technical sessions, a business meeting was held to elect E. G. Lewis of Ajo, Ore Dressing Div. chairman for 1952.

Chairman E. J. Caldwell called the Smelting Div. meeting to order and presented the following technical program: 1. Open discussion on casting large blister copper bars. 2. Presentation of enlarged views of such copper bars as handled at Miami, by Harold Foard, smelter superintendent, Inspiration Smelting & Refining Co., Miami, Ariz. 3. Moving picture of Magma's experimental method of casting large copper bars, by E. J. Caldwell, smelter superintendent of Magma Copper Co., Superior, Ariz. 4. Experiments with large bars by the Hayden smelter, by Frank Downey, smelter superintendent, American Smelting & Refining Co., Hayden, Ariz.

Mr. Caldwell opened the discussions by briefly telling of their experience at Magma with the casting of large copper bars, weighing from 3000 to 3500 lb. These were cast in copper molds weighing about 4500 lb. It was brought out during the discussion that less segregation of precious metals was found to take place in these large bars than in the smaller bars that are usually cast for shipment to the refinery. Mr. Foard's pictures enabled the group to obtain a better idea of this operation. This also can be said of the moving pictures as presented by Mr. Caldwell of their casting technique.

Mr. Frank Downey was unable to attend the meeting so Mr. Groff of the same company discussed their experience with casting of as large bars as was possible with a casting wheel. These were of the order of 700 lb.

A short business meeting followed the technical sessions at which Frank Day of Cananea was elected chairman for 1952.

The cocktail hour was 6:30 pm with dinner at 7:30 pm.

Arizona Section chairman Albert Mendelsohn extended his greetings to the assembled group and the annual report was read by John C. Loving. Each division chairman then presented a brief resume of activities for 1951 and gave the names of the new officers for 1952.

The chairman of the nominating

committee for Arizona Section officers for 1952, C. R. Kuzell, then presented the list of candidates who were unanimously elected. They were W. P. Goss, Magma Copper Co., Superior, Ariz. as chairman; Lyle M. Barker, Phelps Dodge Corp., Morenci as first vice-chairman; J. A. Richards, St. Anthony Mining & Development Co., Tiger as second vice-chairman; and John F. Buchanan, Magma Copper Co. as secretary-treasurer.

R. E. O'Brien, Western Mining Branch Secretary AIME was next introduced by Mr. Mendelsohn. Mr. O'Brien briefly discussed Institute and student affairs.

Mr. Mendelsohn then introduced President Richard Harvill of the University of Arizona who extended his greetings on behalf of the University and an invitation to visit the campus.

The main speaker and honored guest of the evening was W. R. Mathews, editor of the Arizona Daily Star, who spoke on *Report on Europe*.

Minerals and Metals Advisory Board Organized

A new Minerals and Metals Advisory Board has been organized under the National Academy of Sciences-National Research Council. This Board will serve as a research advisory group to the Dept. of Defense and to civilian governmental agencies responsible for research and development in mineral technology and metallurgy for the defense effort.

Dr. Francis C. Frary, recently retired director of research and now technical advisor of the Aluminum Co. of America, will head the new Board as chairman. Dr. Zay Jeffries, retired vice-president of the General Electric Co., and Dr. Robert F. Mehl, director, Metals Research Laboratory, Carnegie Institute of Technology, will serve as vice-chairmen of the Board.

The new Board will continue advisory service to the Research and Development Board and will have the added responsibility of advising civilian defense agencies through the Director of the Bureau of Mines.

The 1952 Mining Branch cloth-bound annual Transactions volume containing the papers appearing in the MINING ENGINEERING Transactions section during 1952 will carry the Volume number 193. Volumes are released early in the year following the date they carry.

Northwest Conference Program Chairmen Named

Program chairmen for the Pacific Northwest Joint Conference have been announced as follows: R. G. Vervacke, Industrial Minerals Div.; A. Y. Bethune, Extractive Metallurgy Div.; Ray Ward, Physical Metallurgy Div.; J. P. Spielman. Mineral Industry Education Div.; and J. Johnson, Iron & Steel Div. H. P. Sherman is in charge of the hotel and meetings; R. O. Oscarson, finance; and W. S. Peterson, publicity.

A luncheon on Saturday, May 10, will feature a talk on northwest gas and the banquet speaker on Saturday evening will be J. F. Bell, president of the Portland Gas & Coke Co. He will present a nontechnical talk on *Quartz and Mica Mining in Brazil* with color photography illustrations. The Spokane Chapter of the WAAIME is planning a program for the ladies attending.

Student Conclave Planned

On January 25, at a meeting of the Executive Committee of the St. Louis Section the following is one of the items voted:

Voted a revolving fund of $1000, from the gift of 1951 Annual Meeting of Guarantors refunds, to finance an annual student's conclave of our five student affiliates. The first is scheduled for mid-April to be held here. On alternate years thereafter a conclave will be held at each school in turn. It is believed the fund will finance three affairs and that it will become a permanent fixture thereafter. L. P. Davidson was appointed Chairman of the Committee to organize the meeting which, after organization, will be conducted entirely by the students themselves.

AIME dues are payable January 1 of each year. Copies of the journals are sent regularly for the first three months of the new year pending receipt of dues. If dues are not received at AIME headquarters by the middle of March, or when the wrappers for the April issues are run off, the member's stencil is removed from the circulation list, and there will be delay in receipt of the April issue. When dues are finally paid, back issues will be supplied to the extent only that copies are available. Duplicate bills should be mailed early in March to those in arrears, and prompt payment at that time, or before, is urgently requested. Resignations, to be accepted as from a member in good standing, must be received before March 31.

Engineering Societies Take Part in Centennial

Several additional national engineering societies have voted to participate actively in the Centennial of Engineering to be celebrated in Chicago. The convocation program has been scheduled early in September although several features open to the general public are tentatively planned for July inaugurals. Representatives of 45 national and international societies are expected to participate.

Award President's Banner

The winner of the President's Banner for the largest number of new applications for membership in its last quarter of 1951, in proportion to its membership, is the Southwestern New Mexico Section. The Banner was presented by President Peirce at the Welcoming Luncheon on February 18. The second highest percentage was attained by the Southwest Texas Section; third by Carlsbad Potash; and fourth by the newly formed Mexico Section.

Student Chapter Approved

Recognition of a Student Chapter of AIME at the Wisconsin Institute of Technology at Platteville, Wis., was approved at the meeting of the Executive and Finance Committees on January 9. The following are officers of the new chapter: Carl R. Christiansen, Faculty Sponsor; George L. Sullivan, Counselor; and James Webster, Secretary, Student Chapter.

Oak Ridge Offers Courses

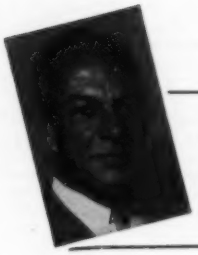
Three courses in the techniques of using radioisotopes in research will be offered by the Special Training Div. of the Oak Ridge Institute of Nuclear Studies this summer. Dates for the courses are: June 9 to July 4, July 7 to August 1, and August 11 to September 5.

These courses are designed to acquaint mature research workers with the safe and efficient use of radioisotopes in research. Each course is open to 32 participants.

Application blanks and additional information may be obtained from Ralph T. Overman, chairman, Special Training Div., Oak Ridge Institute of Nuclear Studies, Oak Ridge, Tenn.

Staff Appointments Made

At the meeting of the Executive and Finance Committees, H. Newell Appleton, as head of the newly organized Institute Activities Dept., was made an Assistant Secretary of the Institute instead of Assistant to the Secretary, his former title. John V. Beall was appointed as Manager of Publications of the Institute to succeed Thomas E. Lloyd whose resignation was effective February 1. Mr. Beall will continue as Editor of MINING ENGINEERING.



THE DRIFT OF THINGS

by Edward H. Robie

THOUGH the final figures are not available as this is written, it seems certain that the Institute accounts will show $7000 to $8000 surplus for the year 1951. This is the first time this has happened since 1941. No money had to be taken from any of the Funds to bring about this happy condition. Furthermore, the amounts allowed for depreciation, bad debts, and inventory adjustment were more liberal than in the past.

Income increased some $29,000 over 1950. Dues received from old members increased moderately, but dues from new members dropped off $2000 and from initiation fees quite severely—from $17,000 to $11,000. The number of new members did not come up to expectations and a change in procedure caused a delay in getting some of them on the rolls. A large number of the new members paid an installment of $5 on their initiation instead of the full $20. Junior Members, of course, pay no initiation fee.

Brightest spot in the income picture was the increase in advertising income—from $103,000 in 1950 to $143,000 in 1951. All three of the journals shared in this, Mining being up $18,000; Metals, $9000; and Petroleum $13,000. Nonmember sales of the journals dropped off slightly, and sales of Transactions volumes more so—from $35,000 in 1950 to $27,000 in 1951, with a further drop of $5000 accounted for by a change in accounting procedure. On the other hand, sales of other publications increased by $10,000.

Expenditures in 1951 increased by only about $6000 over 1950, an exceptionally good showing in view of the decrease in the value of the 1935-39 dollar from 58¢ at the end of 1950 to 53¢ at the end of 1951. This decrease means higher prices for the things the Institute has to buy; especially, higher salaries have to be paid for the office help who do the work at Institute headquarters and in its branch offices.

A marked saving in publishing costs was made because no Directory of members was published in 1951. This cost $19,000 in 1950 but the officer and committee list issued last year cost only $2000.

Expenses chargeable to technical Divisions and Branches increased from $39,000 in 1950 to $54,000 in 1951, the increase being chiefly in the Mining Branch. One reason was that the expense of the Salt Lake City office was all met out of operating income in 1951, whereas its cost was taken out of income in the Barron Fund in 1950.

Expense for the three journals dropped $11,000. This is quite surprising considering the high quality that was maintained. It shows particular care on the part of the editorial and publishing staffs in condensation and editing of articles, and avoidance of luxurious printing practices. Costs of both the Mining and Metals journals decreased, but improvement in the Petroleum journal increased its cost about $7000.

Compared with 1950, general and administrative salaries, and accounting and membership department expenses increased from $93,000 to $95,000, a most modest increase in view of the higher salaries that had to be paid. A saving of $8000 was made in the pension account because of staff changes. Miscellaneous expenses increased $5000; the allowance for depreciation of equipment, furniture, and fixtures, including adjustment for prior years, accounted for $12,500 added expense; and a reserve was set up of $3000 for bad debts, an accumulation of some years.

Branchwise, the Petroleum Branch again, as in 1950, operated on practically a balanced budget as between income and expense. The Mining Branch had a surplus of $35,000 compared with $24,000 in 1950; and the Metals Branch decreased its deficit from $42,000 in 1950 to $27,000 in 1951.

Emphasis in 1952 must be made on increasing income in two main directions: Increasing the advertising, and getting more new members. The latter not only adds dues and initiation income, but adds to the circulation of the journals thus making them better advertising media. In 1950, 21 pct of the AIME income came from advertising and 63 pct from dues and initiation; in 1951 the respective figures were 28 pct and 59 pct. Contrast these figures with those of a sister society, such as the American Society of Mechanical Engineers for instance, which last year received 41 pct of its income from advertising and only 40 pct from dues and initiation fees, even though it has 37,500 members and its dues, for the Member grade, are $25.

By getting more new members and more advertising the expected continued increase in costs due to inflation may be met. Other avenues of income also will be studied. Publication of a Directory in 1952 will add a substantial item to costs. On the whole, it would seem that a surplus should be shown again in 1952 but it will probably be only a modest one, as it was last year.

It would appear that there is little hope, before 1953, of doing much in the way of rebuilding the Institute's funds available for meeting deficits. They almost reached the vanishing point in 1950.

AIME Fall Meeting in Chicago

Plans have been concluded for the dates and location of the AIME Fall Meeting in Chicago next September. This will be in connection with the Centennial of Engineering celebration, which is to cover the first two weeks of the month. A total of 51 engineering organizations will hold meetings during that fortnight.

AIME technical sessions will be held in the Palmer House on Friday and Saturday, September 5 and 6, with a dinner-dance Saturday night. One or more Divisions, that contemplate an extensive program with field trips, may add Thursday, the 4th, to their meeting dates. Already, the Industrial Minerals Div. and the Minerals Beneficiation Div. are definitely planning to participate; the Mining, Geology, and Geophysics Div. is almost a certainty; and several other Divisions planned to make their decision at the Annual Meeting.

The dates selected will enable those who attend to have Sunday for visiting the Museum of Science and Industry, where many special engineering exhibits will be installed. On Monday and Tuesday following, there will be special convocations on the metals industries, with leading speakers, arranged by the Centennial committee. On Wednesday night there will be a banquet for the entire engineering group, which should attract several thousand people.

It looks as though Chicago should be included in the vacation plans of all engineers in 1952.

Personals

C. J. ABRAMS

C. J. Abrams, general superintendent of operations, Climax Molybdenum Co., Climax, Colo., was named general manager of operations. He will have his headquarters in Denver. **Franklin Coolbaugh** will succeed Mr. Abrams at Climax. **Robert E. Henderson,** formerly mine superintendent, has been reemployed as general superintendent at Climax.

H. I. Altshuler, general manager, Frontino Gold Mines Ltd., Segovia, Colombia, returned recently after a vacation in the United States. **Nick J. Hurley** has taken the position of mine superintendent with the same firm.

Arnold D. Arnaut has accepted a position as sales engineer with the Denver Equipment Co. He will be in the New York office. Mr. Arnaut had been employed by Sylvania Electric Products, Inc., physics laboratory, Bayside, N. Y.

Peter A. Bosher has accepted an appointment as a mining engineer with the Sierra Leone Development Co., Ltd., Sierra Leone, British West Africa.

Denny R. Benner is a surveyor with the St. Joseph Lead Co., Bonne Terre, Mo.

Leroy T. Brown has opened his office as a mining engineer at Gouverneur, N. Y.

Louis Buchman has been elected vice-president and director of Kennecott Copper Corp. Mr. Buchman will be located in Salt Lake City and continue to supervise all of Kennecott's western mining divisions.

Charles E. Basso has become general superintendent for Bolivian Tin & Tungsten Mines Corp., Huanuni, Bolivia.

Lee E. Blinzler is now mining engineer for the General Refractories Co., Philadelphia.

Gordon H. Chambers, executive vice-president, Foote Mineral Co. is making a trip to Sweden, Holland, western Germany, Great Britain, France, and North Africa.

James D. Carter is now preparation engineer for the Capitol Fuel Co., Princeton, W. Va.

Vern H. Christoph has joined the Panama Canal Co. as a surveying and cartographic engineer.

James Douglas, former Defense Minerals Administration deputy administrator has been named assistant deputy administrator, Defense Materials Procurement Administration. **Tom Lyon,** former director of DMA's supply div., has been named director of the domestic expansion div., DMPA. **Charles E. Stott,** former director of ECA's strategic materials div., was named director of the foreign expansion div., DMPA. **A. B. Parsons,** formerly with DMA is now acting director of the program development div., DMPA.

L. Elton Davis is now with the American Ventilating Hose Co., Amesbury, Mass.

John C. Draper is now employed by the Consumers Mining Co., Harmarville, Pa., as an industrial mining engineer.

John T. Eastlick, assistant mining engineer, Anaconda Copper Mining Co., Butte, Mont., has been made stope geologist at Darwin, Calif.

Charles A. Fellencer, Allentown, Pa., is now in Peru for Ebasco Services, Inc. as project manager. He is working on the installation of a hydroelectric plant for Cerro de Pasco Corp.

Herbert J. French, vice-president, International Nickel Co., Inc., presented the Burgess Memorial Lecture before a meeting of the ASM, Washington, D. C.

J. Walter Fulmer is with the American Metal Co., Ltd., New York.

American Smelting & Refining Co. Announces Personnel Changes

W. H. Loerpabel, formerly general manager, mining dept., New York, has been elected a director and executive vice-president of Northern Peru Mining & Smelting Co., and vice-president of Compania American Smelting, Boliviana, S. A. He will direct the company's business in Peru and Bolivia from headquarters in the New York office.

C. P. Pollock, formerly resident engineer, mining dept., New York, has recently been appointed assistant to the vice-president.

L. H. Hart, formerly exploration engineer of the western mining dept., Salt Lake City, has been appointed resident engineer with headquarters in New York.

A. C. Hall has been transferred from Charcas unit where he was superintendent to Lima, Peru as managing director of the Northern Peru Mining & Smelting Co., where he will be in charge of the company's interests in Peru and Bolivia.

Sergei Zelenkov has been transferred from Leadville, Colo., where he was superintendent to Lima, Peru, where he is manager of operations of the Northern Peru Mining & Smelting Co.

E. McL. Tittmann has been transferred from El Paso where he was manager of the southwestern dept., to Salt Lake City where he will be general manager of the western dept.

Ben D. Roberts, formerly assistant manager at El Paso, has been promoted to manager of the southwestern dept.

Frank H. Eichler, formerly manager of the Federated Metals Div., Los Angeles plant, has been transferred to the New York office where he will be director of purchases.

E. H. Scheick has been appointed chief engineer, New York.

C. M. Syner, formerly superintendent of the Santa Eulalia unit, has been transferred to the Charcas unit as superintendent.

C. E. Nelson, formerly general manager of Neptune Gold Mining Co. has been transferred to the Buchans Mining Company, Ltd. where he will be general manager, succeeding **George G. Thomas** who has retired.

L. J. Siessinger has been transferred from the San Carlos unit where he was superintendent to the Nuestra Senora Unit, Cosala, Sinaloa, where he will act as superintendent.

M. C. Hirtzel, formerly assistant superintendent of the Santa Eulalia unit has been promoted to superintendent.

T. A. Snedden is now superintendent of the Silver Bell unit.

G. L. Fischer has been appointed superintendent of the San Carlos unit.

Allison J. Gould has become mine superintendent, Yerington mine, Anaconda Copper Mining Co., Yerington, Nev.

Ellis H. Gates has resigned as director of research with the Cerro de Pasco Corp., La Oroya, Peru. He is returning to Long Beach, Calif.

Joseph R. Gross is now associated with the Inter-State Iron Co., Virginia, Minn.

Merle H. Guise has joined the Mineral Products Corp., Rockford, Ala.

Guy E. Giroux is president, Red Fern Corp., Robinson, Ill.

Donald W. Hammerquist has accepted a position with the South Dakota Highway dept., Pierre, S. D.

Lewis J. Hash is presently employed as a product engineer with the General Electric Co., Coshocton, Ohio.

Alvin P. Humphries is now with the Tennessee Coal, iron div., U. S. Steel Co., coal div., Birmingham, Ala., as an industrial engineer.

Paul A. Hodges is now employed as a junior engineer at the Silver Bell mine in Tucson, for the American Smelting & Refining Co. He recently completed his training period.

Verdo Z. Johnson is with the Bureau of Mines at Boston. He had been located in Platteville, Wis.

Elmer A. Jones has become assistant general manager of the Southeast Missouri Div., St. Joseph Lead Co. He had been assistant general mine superintendent.

F. R. Jeffrey has been made manager of the ore dept. of International Minerals & Metals Corp., and treasurer and assistant to the president of National Zinc Co., Inc., New York.

D. F. Kidd is the newly appointed president of the Canadian Institute of Mining & Metallurgy.

John B. Knaebel, Anaconda Copper Mining Co., New York, has been transferred to Grants, N. Mex.

Victor E. Lovejoy is manager, Mineral Products Corp., Chewelah, Wash.

Jack C. McFarlin is now employed as a junior engineer by the Reynolds Mining Corp., Alexander, Ark.

A. E. Millar is manager, Yerington mines, Anaconda Copper Mining Co., Yerington, Nev.

Lowell B. Moon, chief of the minerals div., U. S. Bureau of Mines, will join the exploration staff of Kennecott Copper Corp.

Benjamin F. McGuire has been made mine surveyor for the Duval Sulphur & Potash Co., Carlsbad, N. Mex.

Elwood S. Moore, professor of economic geology, University of Toronto (retired) recently gave a series of lectures at Columbia University. **J. T. Singewald, Jr.,** professor of economic geology at Johns Hopkins University, also gave three lectures.

Thomas G. Murdock, American counsel and minerals officer at Elisabethville, Belgian Congo, was recently in the United States visiting Washington, North Carolina, and Texas.

Thomas Wilson Mitcham is geologist for the S. W. div., American Smelting & Refining Co., Tucson.

Wallace W. Muehl has been made superintendent of the crushing and Sink Float plant, Patino Mines & Enterprises, Llallagua, Bolivia.

John F. Myers has retired as superintendent of concentration, Tennessee Copper Co., Copperhill, Tenn. **F. M. Lewis** will succeed Mr. Myers, who will be retained in a consulting capacity. Mr. Myers is an AIME director and was the recipient of the Robert H. Richards Award for 1952.

Richard A. Matuszeski has accepted a position as supervisor in the metallurgical dept. of the National Lead Co. of Ohio. He had been a senior engineer in the tungsten and chemical div. of Sylvania Electric Products, Inc., Towanda, Pa.

G. H. W. Meeder is now employed by the Chile Exploration Co., Chuquicamata, Chile.

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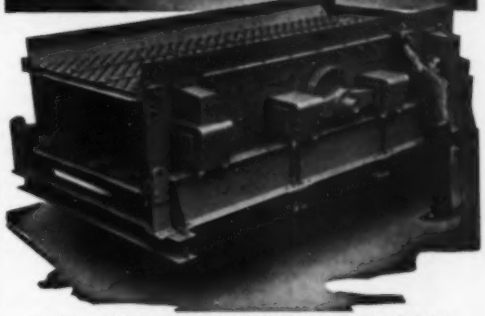

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J. B. MORROW

J. B. Morrow has retired as a director of Pittsburgh Consolidated Coal Co. and will form the engineering consulting firm of Alford, Morrow & Associates, Pittsburgh, with **N. G. Alford,** as successor to Newell G. Alford & Associates. Mr. Morrow is an AIME director and Mr. Alford a past director.

Alfred E. Nugent is now a geologist for the Bunker Hill & Sullivan Mining Co., Kellogg, Idaho. Mr. Nugent had been manager of the N & N Mining Co., Helena and president of Last Chance Gulch Mining Assn., Helena.

Peter Borie Nalle has resigned from the St. Joseph Lead Co. and joined the Riverside Cement Co., Riverside, Calif.

Robert S. Palmer, executive director of the Colorado Mining Assn., has been appointed consultant to the Defense Materials Procurement Agency.

Thomas B. Piper has accepted a position as geologist with the Atomic Energy Commission, Denver. He had been with the E. J. Longyear Co., Minneapolis.

Hans Pick is now employed as mechanical engineer by the Compania Minera Aguilar, La Paz, Bolivia. He had been associated with the Compagnie Aramayo de Mines.

GUSTAV S. PRELLER

Gustav S. Preller, mineral technologist, African Metals Corp., Johannesburg, South Africa, has been appointed manager of Phosphate Development Corp., Pretoria, South Africa. The firm has been formed by the government of the Union of South Africa for the purpose of producing apatite concentrates.

Joseph S. Quinn, sales representative, Allis-Chalmers Mfg. Co., has been transferred from Milwaukee to St. Louis.

Lewis Roberts is now a geologist with the Atomic Energy Commission, Grand Junction, Colo.

William G. Richards, Ingersoll-Rand Co., is now at the New York office. He had been located in Salt Lake City.

Miles P. Romney, former executive vice-president of the Duvall Co., Cassia, Idaho, and geologist for the U. S. Smelting, Refining & Mining Co., Salt Lake City, was named secretary-manager of the Utah Mining Assn.

W. G. Rouillard, plant manager, American Smelting & Refining Co., has been elected president of the Utah Mining Assn. Mr. Rouillard is also chairman of the Utah Section, AIME. **R. D. Bradford** has resigned as president of the Utah Mining Assn. He was transferred to the New York office of American Smelting & Refining Co.

George W. Rust has accepted a position with Kennecott Copper Corp., New York, as geologist.

Vincent G. Rumpf is now a mine engineer with Telluride Mines, Inc., Telluride, Colo.

CHESTER H. STEELE

Chester H. Steele, head of the Butte mines, Anaconda Copper Mining Co., has been made general manager of Western mining operations. **Edward P. Shea,** mines geologist, was named to succeed Mr. Steele. **A. S. Richardson** has retired as chief ventilation and industrial hygiene engineer for western operations. He will remain as a consultant. **Martin K. Hannifan** had been appointed assistant general superintendent of mines in charge of the Kenney mine.

R. L. Scott has been promoted to assistant to general manager, commercial steel sales, western div., Colorado Fuel & Iron Co., Denver.

Stuart St. Clair recently returned from southeast Asia. He had been examining mineral deposits and mining activities.

Richard W. Smith, assistant manager of the National Chamber of Commerce Natural Resources Dept., has been named department manager, succeeding **David J. Guy,** who recently retired.

Woodrow W. Slade, Owens-Illinois Glass Co., has been transferred from research on raw materials in Toledo, Ohio, to the west coast general engineering dept., Oakland, Calif. He is project engineer for all raw materials.

Frank Seery, project engineer for the General American Transportation

L. F. Pett, general superintendent of mines, Kennecott Copper Corp., Utah Copper Div., has become general superintendent of operations, with headquarters in Salt Lake City.

J. C. Landenberger, Jr., of Salt Lake City, director of labor relations, has been made general superintendent of mines, succeeding Mr. Pett.

Joseph A. Norden, Jr., assistant general drilling and blasting foreman, is now director of labor relations, succeeding Mr. Landenberger.

V. S. Barlow, assistant mine superintendent, has been promoted to mine superintendent.

James K. Richardson, of Salt Lake City, industrial engineer for Kennecott's western mining div., was named assistant to the Utah Copper Div. general manager in charge of public, industrial and labor relations. He succeeds **Roy Hatch** who retired.

Corp., is now in Pittsburgh. He had been at the Chicago office.

Peter R. Stevens is now with the U. S. Geologic Survey, Holbrook, Ariz.

J. Van den Berg has joined the National Lead Co., New York as mining engineer.

FORBES K. WILSON

Forbes K. Wilson has been appointed manager of mineral exploration for Freeport Sulphur Co.

Otto Witte has resigned from Diamond Products, Inc., Elyria, Ohio.

Roman H. Wellington has joined the Kennecott Copper Corp., Ruth, Nev.

Louis P. Weyand was promoted to executive vice-president, Minnesota Mining & Mfg. Co., St. Paul. Associated with the firm for 37 years, Mr. Weyand was elected a vice-president in 1946 and named a director in 1950.

John Pershing Wolfe is now employed by the Magma Copper Co., Superior, Ariz., as a miner.

Langbourne M. Williams, Jr., president of Freeport Sulphur Co., was elected a trustee of Atlantic Mutual Insurance Co. and a director of Centennial Insurance Co.

Fred Wartenweiler has retired as consulting metallurgist for Johannesburg Consolidated Investment Co., Johannesburg, South Africa.

James S. Wroth has resigned as executive vice-president and director of the International Mining Corp. of New York. Mr. Wroth has been associated with the company since it was organized in 1928.

Robert Charles Youtz is now quarry engineer for the U. S. Gypsum Co., Southard, Okla.

Paul Zinner, regional director of Region V, Minneapolis, has been named to the post of chief of the minerals div., U. S. Bureau of Mines, Washington, D. C. **Paul T. Allsman,** who has been at the Denver office, will succeed Mr. Zinner.

Obituaries

H. P. Banks (Member 1919) has died. Mr. Banks had been employed by I. F. Lauks, Inc., Seattle, for many years. In 1919 he was chief chemist, in 1927 was made vice-president and manager, and in 1940 was vice-president and general manager. At the time of his death he was residing in Seattle.

H. A. Barker (Member 1902) of Santa Fe, N. Mex., has died. Mr. Barker was born on Apr. 16, 1871 in Scotland. For six years he was an apprentice pupil and assistant at the City Analysis Laboratory, Glasgow, and did special work at the Glasgow and West of Scotland Technical College. In 1893 he was chemist for the British Hydraulic Foundry Co. At this time he was working on the Cyanide process. For approximately three years he was chemist and assayer for the South African Republic. Mr. Barker came to the United States and was engaged in demonstrating and applying the Cyanide process to gold ores in the western states. He was employed by the McArthur & Forrest Process Co., Denver and was later made superintendent of Cyanide works for King of Arizona. He joined the Free Gold Mining Co., Hidjrs, Calif. In 1918 Mr. Barker went to Peru as general manager for the Sociedad Esplotadora de Caylloma Consolidated. He returned to Scotland and England for a short time, and then did examining, valuing and consulting at mines in Brazil, Chile, Argentina, Bolivia, and Peru. In 1928 he was employed by the Dorr Co., New York. Four years later he moved to Dallas and in 1936 was located in New Mexico.

John J. Clifford (Member 1936) died on Mar. 15, 1951. Born at Denver on Oct. 9, 1895, Mr. Clifford attended the Colorado School of Mines from 1913 to 1917. During World War I he was in the Army and following his discharge was a surveyor for the U. S. Reclamation Service. In 1920 he joined the San Luis Mining Co., Tayoltita, Mexico as mine surveyor. Mr. Clifford was promoted to assistant mine superintendent in 1922. He went to Mexico in 1924 as mine foreman for the Mazapil Copper Co. in Aranzazu. In 1926 he joined the Cia. Real del Montey Pachuca, Hidalgo, as mine foreman and in 1930 was made mine superintendent. Mr. Clifford remained in Mexico for many years, but at the time of his death was residing in Denver.

Alvin M. Cummings (Member 1916) died of a heart ailment on Jan. 6, 1952. Mr. Cummings was born at Lyon Mountain, N. Y. in 1886. He received the degrees of B.S. and E.M. from the Michigan College of Mines in 1912. After graduation he joined the Witherbee Sherman Corp. and was engaged in surveying, concentration, exploration, and construction of plant buildings. In 1916 Mr. Cummings was promoted to mine superintendent and in 1919 became general superintendent for the firm. He was later appointed to the position of general manager. In 1939 Mr. Cummings joined the Atlas Feldspar Co. He worked on examinations and reports on New York and New England feldspar mines and mills. In 1940 he joined the U. S. Bureau of Mines as project engineer at Arkansas, West Virginia, and South Dakota. He also was located in Wisconsin and Lake Superior for the Bureau. Four years later he became associated with the International Talc Co. and Oswegatchi Light & Power Co. as general manager. For the past three years, Mr. Cummings had been with the Corps of Engineers, Pickstown, S. D., working on the Fort Randall Dam project.

Oscar A. Dingman (Member 1936) died during December 1951 after having been in ill health for several months. Born in Aurora, Mo., Nov. 14, 1889, Mr. Dingman received his de-

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gree of B.S. in mining engineering at the University of Kansas in 1914. After graduation he was a millman for the Chanton Gold Mining Co., La Pas, Honduras. He was a timberman for the Consolidated Mining Co., Goldfield, Nev. In 1915 he came to Butte for the East Butte Copper Mining Co. In succession he became a surveyor, shift boss, and assistant foreman. He enlisted in the Army during World War I and held a captaincy at his discharge. From 1918 to 1923 he was engineer and assistant foreman for the East Butte company and then mine foreman at Kimberly, Nev. for the Consolidated Copper Mining Co. For one year he served as foreman of the Jardine Mining Co. and again returned to the East Butte firm as acting assistant manager. Mr. Dingman began his teaching career at the Montana School of Mines in 1927. He obtained his engineer of mines degree in 1931 and was named professor of mining in about 1935. In 1946 he was elected to the school board and the next year was named chairman of the board. In 1949 he was elected for another term. He was author of bulletins and various technical papers.

B. G. Klugh (Member 1907) has died. Born at Coronaca, S. C. on Nov. 13, 1880, Mr. Klugh attended the South Carolina College from 1897 to 1899 and the University of North Carolina from 1899 to 1901. Following graduation he joined the Sparta Iron Co. In 1905 he joined the Pioneer Iron Co., Marquette, Mich. as chemist. For one year he was employed by the Carnegie Steel Co., Youngstown and in 1907 joined the Youngstown Sheet & Tube Co. as chemist. Mr. Klugh became associated with the Southern Manganese Corp., Anniston, Ala. In 1924 he was secretary and chief engineer for the American Ore Reclamation Co., Chicago. Two years later he was located in Birmingham in the same capacity for the company. He returned to the Southern Manganese Corp. in 1927 as vice-president. In 1932 he was vice-president for the Swann Chemical Co. in Birmingham and then at Anniston. At the time of his death he was with the Monsanto Chemical Co., Anniston.

Thomas F. Maddick (Member 1939) died suddenly at Larkspur, Calif. on Nov. 8, 1951. He was born at Cleveland in 1888. Mr. Maddick attended colleges in England. For three years he was a machinist's apprentice for the Aber Steel Works, Swansea, England. In 1908 Mr. Maddick joined the British Metals Extraction Co. as a machinist, and in 1912 was promoted to plant foreman. Two years were spent in Sweden as master mechanic and mill superintendent and he then returned to the British Metals Extraction Co., London. From 1919 to 1921 he attended the University of Wales. After attending the University he became research engineer for the Sanguhar & Kirkconnell Collieries, Scotland. Mr. Maddick then came to the United States and joined the St. Joseph Lead Co., Bonne Terre, Mo. He joined the Anaconda Copper Mining Co., and at the time of his death was employed in the mechanical engineering dept.

C. R. Millar (Member 1925), London, England, died in June 1951. Mr. Millar was born at New South Wales in 1874 and attended the Camborne School of Mines, England. In 1903 he joined Witwatersrand Deep Ltd., Transvaal. For one year he was employed in Manila, P. I. by the Abucay Estates, Inc. Mr. Millar was on the Gold Coast for approximately 13 years. He joined the African Manganese Co. Ltd. as miner. He was made superintendent of native laborers, surveyor, prospector, mine superintendent, and manager, respectively. He returned to England in 1927 for one year and again in 1931 to remain.

James H. Morley (Member 1922), superintendent of the American Rutile Co., Roseland, Va., died on Apr. 1, 1951. He attended Pennsylvania Military College and received the degree of B.S. in mining from Massachusetts Institute of Technology. He was employed by the Susquehanna Coal Co. and Eustis Mining Co. In 1912 he joined the American Smelting & Refining Co. at various units in Mexico. He returned to the United States and joined the Army during the first world war. In 1919 he was assistant engineer for the Minnesota State Highway Dept. and in 1920 joined the American Rutile Co.

F. B. Nold (Member 1908) has died. Mr. Nold was born in 1882 at Columbiana, Ohio and attended the Ohio State University, School of Mines. In 1905 he received an E.M. degree and was then employed as a transitman. He joined the Hoover & Kinnear Co. as assistant engineer and in 1906 was assistant engineer for the U. S. Coal & Oil Co., Holden, W. Va. Mr. Nold went to Mexico for the Mexico Coal & Coke Co. for two years. For 1½ years he was superintendent of mines and coke works for the northern coal mines of Republic Iron & Steel Co. In 1926 he was district superintendent for the Lehigh Coal & Navigation Co., Lansford, Pa. Later he joined the Edison Anthracite Coal Co. as colliery superintendent. At the time of his death he was residing in Lansford, Pa.

Benjamin F. Tillson (Member 1905), retired head of the mining dept. and assistant superintendent of the N. J. Zinc Co., died suddenly on Dec. 4, 1951. Born in Norwich, N. Y., Apr. 11, 1884, Mr. Tillson received his bachelor of philosophy degree from Yale Uni-

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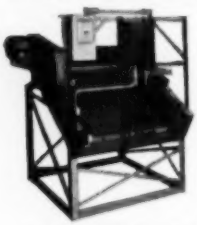

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versity in 1905. Two years later he received his mining engineering degree from Columbia University. He joined the N. J. Zinc Co. following graduation from Columbia and remained with the firm until his retirement in 1950. During the first world war he represented the AIME as a technical consultant on the technical advisory committee of the Office of Civilian Defense for the New York area. He served as chairman of the advisory board of the National Safety Council and of the U. S. Bureau of Mines. During the thirties Mr. Tillson was named as special consultant to the association of contractors for the Lincoln and Queens midtown tunnels. After his retirement from the N. J. Zinc Co. he served as a private consulting engineer.

Albert L. Toenges

An appreciation by H. P. Greenwald

Albert L. Toenges, chief of the bituminous coal mining section of Region VIII of the U. S. Bureau of Mines died at his home in the Pleasant Hills district of Pittsburgh, Pa. on Dec. 19, 1951. He was in his 63 year.

After graduating from the Colorado School of Mines in 1912, he was employed by the Oliver Mining Co. as a mining engineer at Hibbing, Minn., for two years. Then followed three years with the St. Joseph Lead Co. and Missouri Metals Corp. in Missouri, and two years of work in the State of Virginia in the vicinity of Roanoke. In 1912, he joined the staff of the Western Coal & Mining Co. (Missouri Pacific Railroad interests) and remained with that company 16 years on assignments in Missouri, Arkansas, Kansas, and Texas.

In 1935, Mr. Toenges joined the staff of the Bureau of Mines working on bituminous coal mining problems and became chief of that section 7 years later. He participated in and directed numerous studies on mining methods and costs resulting in more than 25 publications.

In recent years he was closely associated with the development of the coal resources of Alaska and was in charge of the Bureau's survey of the reserves of coking coal in the United States. In 1944, Mr. Toenges was selected to make a special study of coal mining in Chile; this was in response to a request from the Chilean Government. In 1949, he made a study of the lignite reserves of Greece for the Economic Cooperation Administration.

In addition to being a member of AIME for more than 35 years, Mr. Toenges was a member of Tau Beta Pi National Engineering Honorary Fraternity. He was a man of high integrity, sound judgment and firmness of character. The mining engineering profession in general and the Bureau of Mines in particular has lost a valuable member replacement of whom will be no easy task.

NECROLOGY

Date Elected	Name	Date of Death
1919	H. P. Banks	Unknown
1902	H. A. Barker	Unknown
1936	J. J. Clifford	Mar. 15, 1951
1916	Alvin M. Cummings	Jan. 6, 1952
1938	J. Eppelsheimer, Jr.	1950
1907	B. G. Klugh	Unknown
1951	Fred H. Loftus	1951
1939	T. F. Maddick	Nov. 8, 1951
1906	John M. Moubray	Oct. 4, 1951
1902	Robert W. Pringle	Unknown
1949	Ernest F. Pullen	Unknown
1942	P. W. Racey	Sept. 20, 1930
1930	William B. Rose	Nov. 10, 1951
1927	Ernest Arthur Satchell	Unknown
1921	Sibyl Walker Smith	Dec. 7, 1951
1919	Raleigh P. Trimble	Jan. 9, 1951
1940	Donald K. Weaver	Unknown
1936	H. Comer Wolf	Oct. 28, 1951
1927	Theodor Zuschlag	Jan. 14, 1952

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Total AIME membership on Dec. 31, 1951 was 17,462; in addition 2229 Student Associates were enrolled.

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Bisbee—Mattingly, Kenneth L. (M) (R. C/S—S-M)
Bisbee—Pincock, Warren K. (M) (R. C/S—S-M)
Clifton—Mann, Robert E. (M) (C/S—A-M)
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Superior—Hall, Arthur B. (M)
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Tucson—Whittemore, Ralph C. (A)
Warren—Metz, Harry E. (M) (R. C/S—J-M)
Warren—Phillipi, Fred L. (A)
Warren—Torgersen, Edward H. (J) (C/S—S-J)

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Menlo Park—Beatty, William B. (M) (R. C/S—S-M)
Oakland—Glenn, George G. (M)
Placerville—Hansen, Lawrence S. (J) (R. C/S—S-J)

Colorado
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Denver—Steele, Robert A. (J) (C/S—S-J)
Denver—Wick, Keith E. (J) (C/S—S-J)
Gilman—Brown, James M. (A) (C/S—S-A)
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New Canaan—Schrader, Walter C. (J)

Florida
Pierce—Goin, Charles M. (J) (R. C/S—S-J)

Idaho
Osburn—Crandall, Wallace E. (M)
Osburn—Gordon, Robert Glenn (J)
Wallace—Randall, Lester J. (A)

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Hinsdale—Koechlein, Frederick A. (M)
Oak Park—Gerler, Warren C. (M)
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South Virginia—Stephenson, Thomas E. (J)
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Ruth—La Prairie, Jules P. (J)

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Tenafly—Grose, Harris W. (A)
Wortendyke—Ross, Hamilton M. (M)

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Kingston—Reid, Edward D. (M)

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Pottsville—Martin, Herbert H. (M)
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Coming Events

Mar. 3-7, ASTM, spring meeting and committee week, Hotel Statler, Cleveland.

Mar. 4, AIME, Morenci Sub-Section, Longfellow Inn, Morenci, Ariz.

Mar. 5, AIME, Chicago Section, Ladies' Night, Chicago.

Mar. 10-13, National Electrical Mfrs. Assn., spring meeting, Chicago.

Mar. 10-14, National Assn. of Corrosion Engineers, annual convention, Galveston.

Mar. 11, Society for Applied Spectroscopy, 6 pm, supper, Tosca's; 8 pm, meeting, Socony-Vacuum Training Center, 63 Park Row, New York.

Mar. 16-19, American Institute of Chemical Engineers, Atlanta Biltmore Hotel, Atlanta.

Mar. 17-21, ASTE, annual meeting, Chicago.

Mar. 20, AIME, Philadelphia Section, Engineering Bldg., University of Pennsylvania, Philadelphia.

Mar. 22-Apr. 6, Chicago International Trade Fair, Navy Pier, Chicago.

Apr. 1, Society for Applied Spectroscopy, Socony-Vacuum Training Center, 63 Park Row, New York.

Apr. 7-9, American Society of Lubrication Engineers, annual meeting, Statler Hotel, Cleveland.

Apr. 9, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Apr. 11-12, Geological Society of America, Cordilleran Section, annual meeting, University of Arizona, Tucson.

Apr. 21-22, American Zinc Institute, annual meeting, Hotel Statler, St. Louis.

Apr. 21-23, Chemical, Metallurgical, and Mining Society, and **Diamond Drilling Research Laboratory**, symposium on diamond drilling, Johannesburg, South Africa.

Apr. 22, Assn. of Consulting Chemists & Chemical Engineers, Inc., general symposium, Hotel Belmont Plaza, New York.

Apr. 25-26, AIME, New England Regional Conference, Kenmore Hotel, Boston.

Apr. 26-May 11, Liege International Fair, Liege, Belgium.

May 1-7, International Foundry Congress, Convention Hall, Atlantic City, N. J.

May 5-6, National Air Pollution Symposium, Huntington Hotel, Pasadena, Calif.

May 5-7, American Mining Congress, coal convention, Netherland Plaza Hotel, Cincinnati.

May 6-9, Scientific Apparatus Makers Assn., annual meeting, Edgewater Beach Hotel, Chicago.

May 9-10, AIME, joint meeting, Industrial Minerals Div. and Metals Branch, Davenport Hotel, Spokane, Wash.

May 11-14, American Institute of Chemical Engineers, Atlanta Biltmore Hotel, Atlanta.

May 22-24, American Society for Quality Control, annual convention, Onondaga County War Memorial, Syracuse, N. Y.

June 16-20, American Electroplaters Society, Industrial Finishing Exposition, International Amphitheater, Chicago.

June 23-27, ASTM, 50th anniversary meeting, Hotel Statler, New York.

July 1-Sept. 30, Centennial of Engineering, Chicago.

Sept. 10-12, Porcelain Enamel Institute, annual shop practice forum, University of Illinois, Urbana.

Sept. 11-13, American Institute of Chemical Engineers, Palmer House, Chicago, Ill.

Sept. 22-25, American Mining Congress, Metal and Nonmetallic Mining Convention and Exposition, public auditorium, Denver.

Oct. 22-24, Porcelain Enamel Institute, annual meeting, Greenbrier Hotel, White Sulphur Springs, W. Va.

Nov. 19, American Mining Congress Coal Div. Conference, Wm. Penn Hotel, Pittsburgh.

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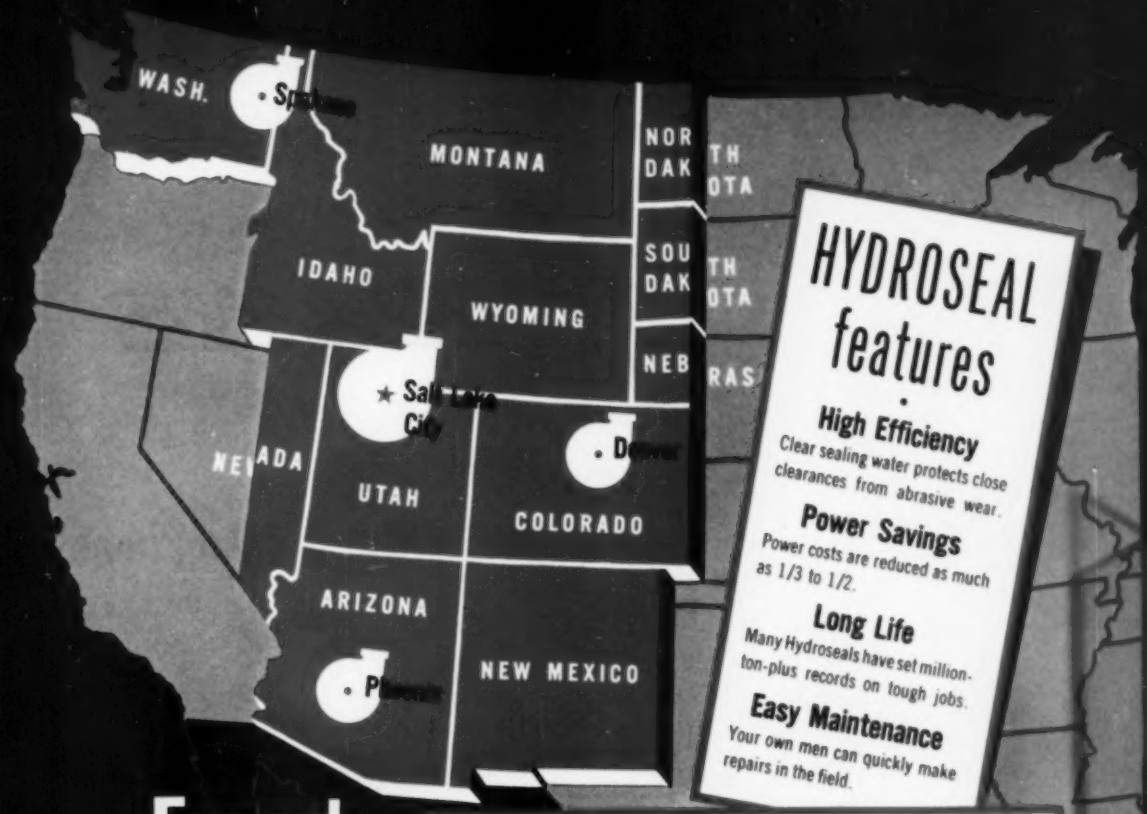

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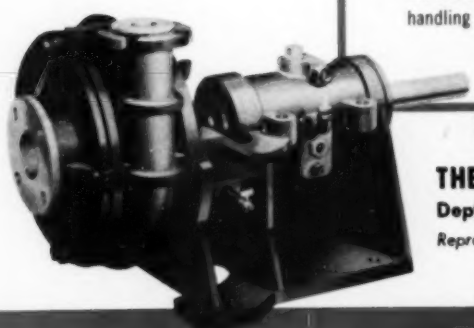

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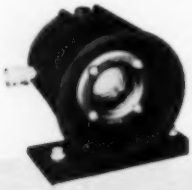

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